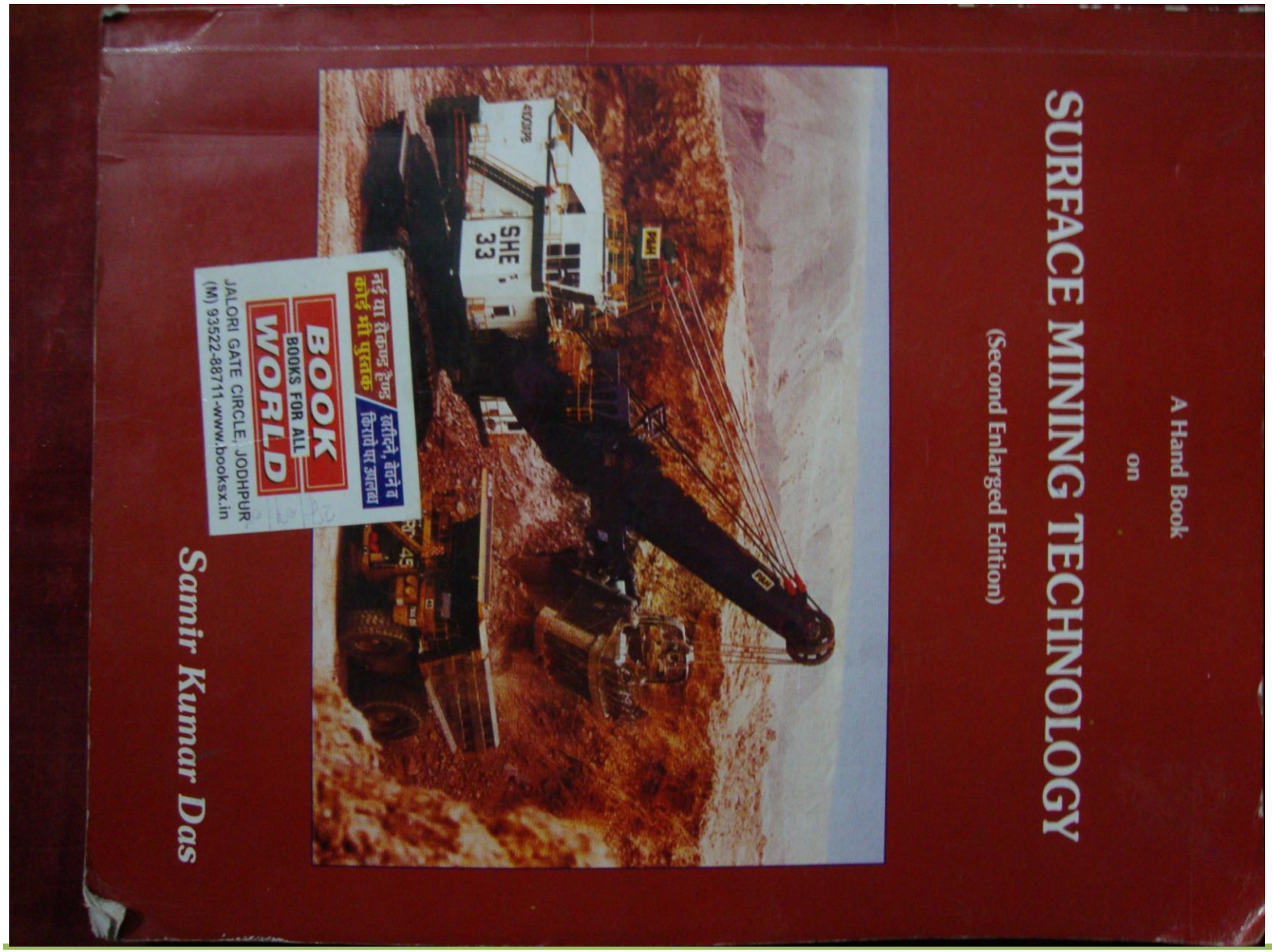

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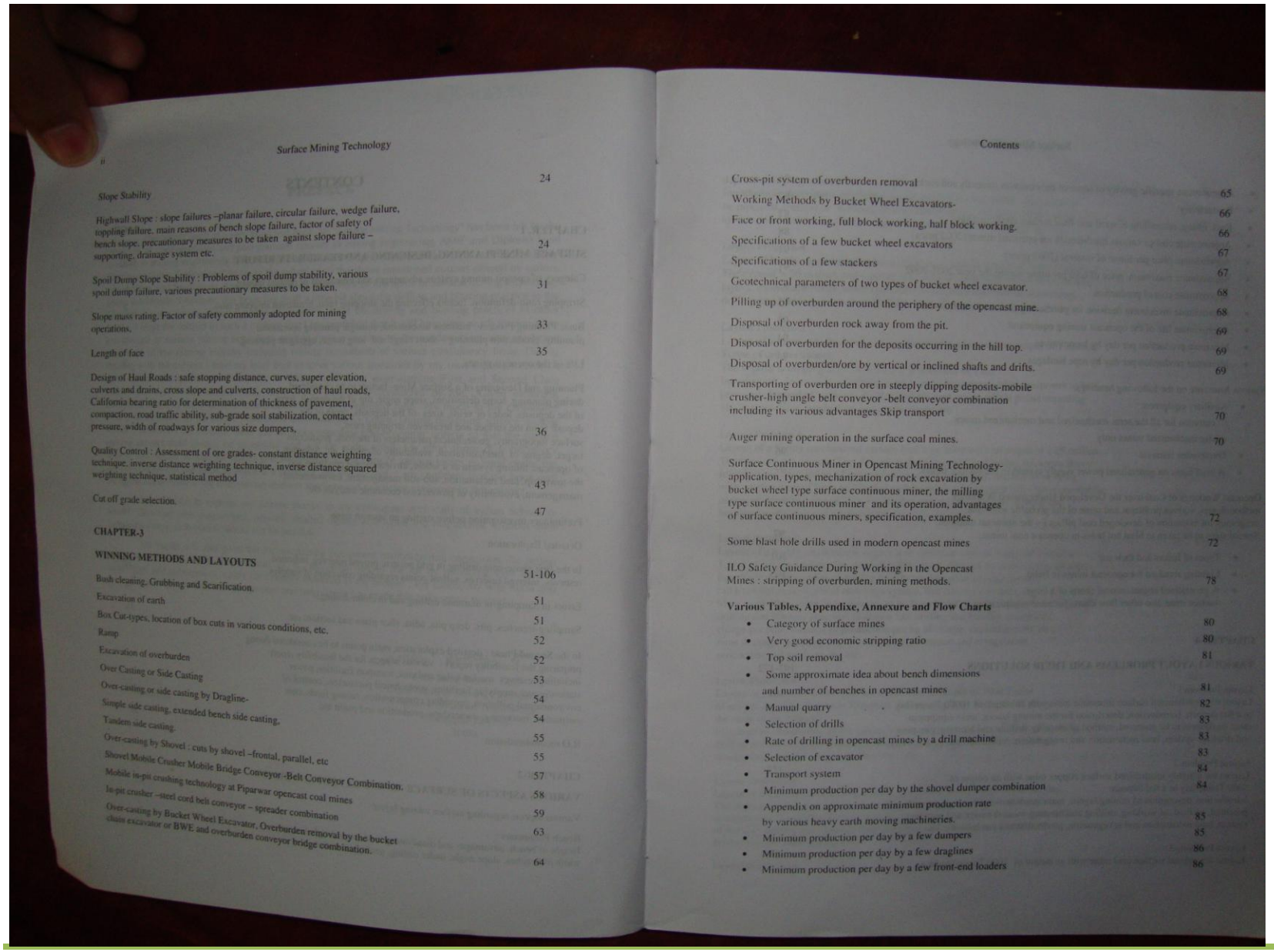

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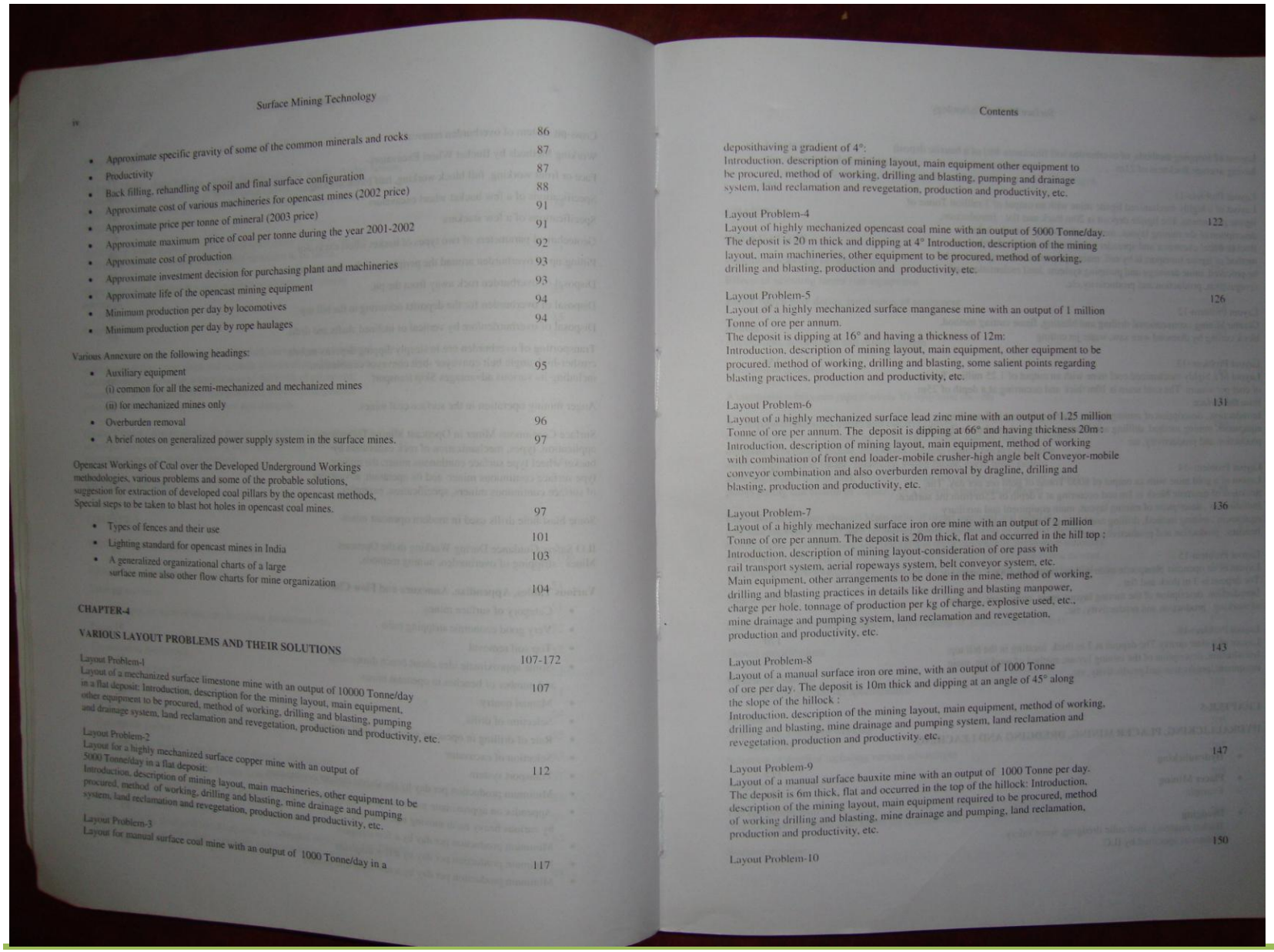

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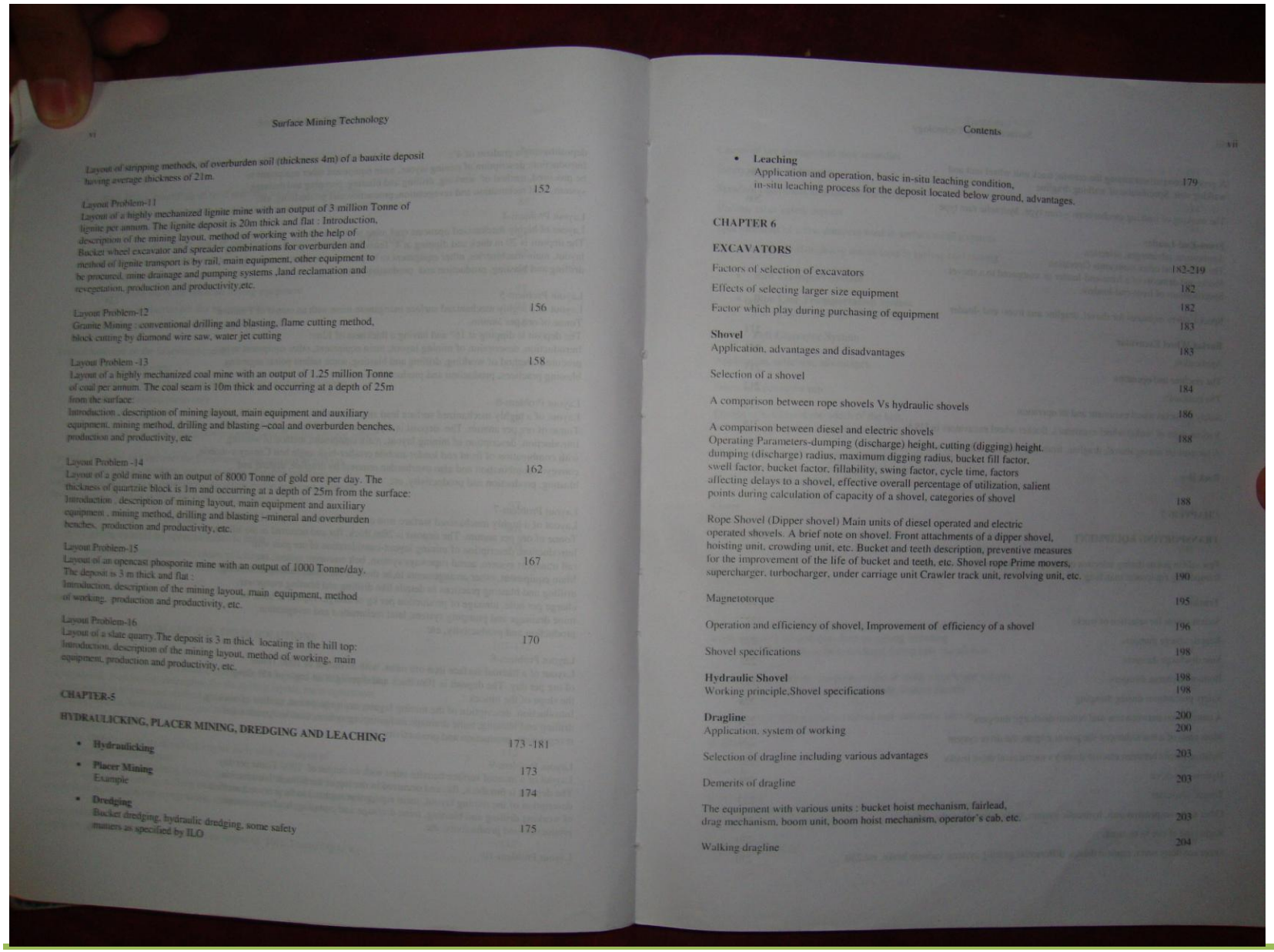

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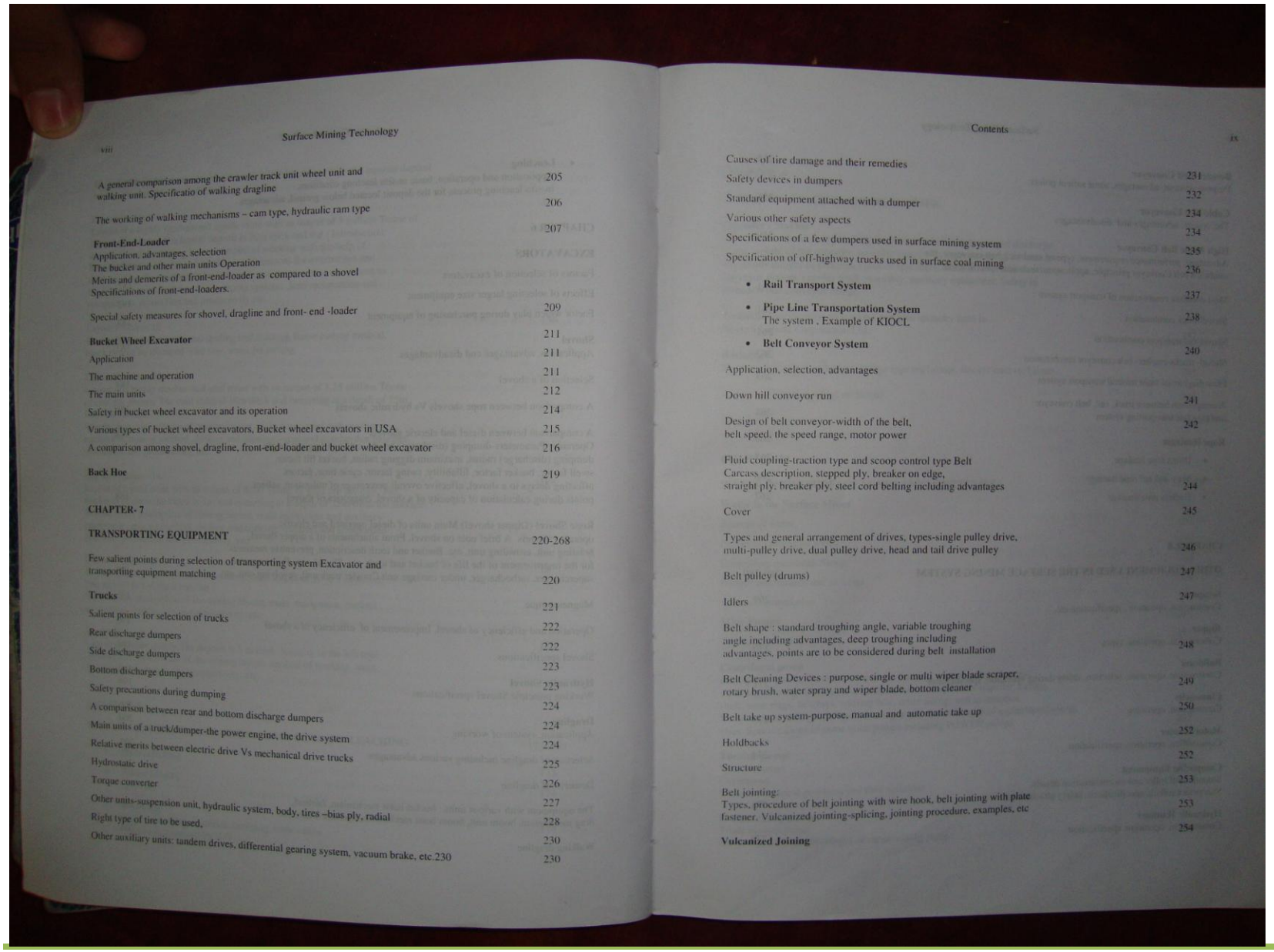

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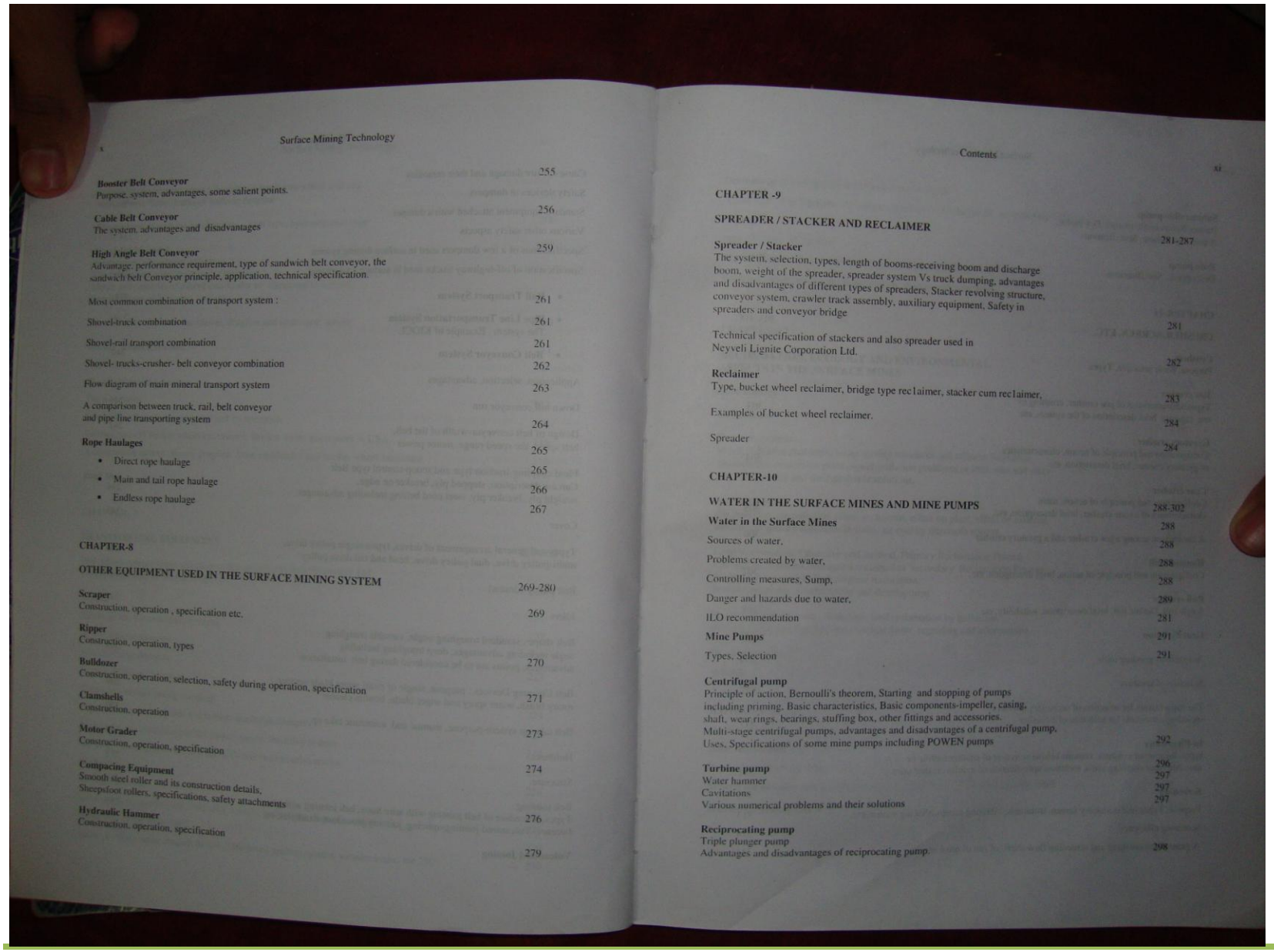

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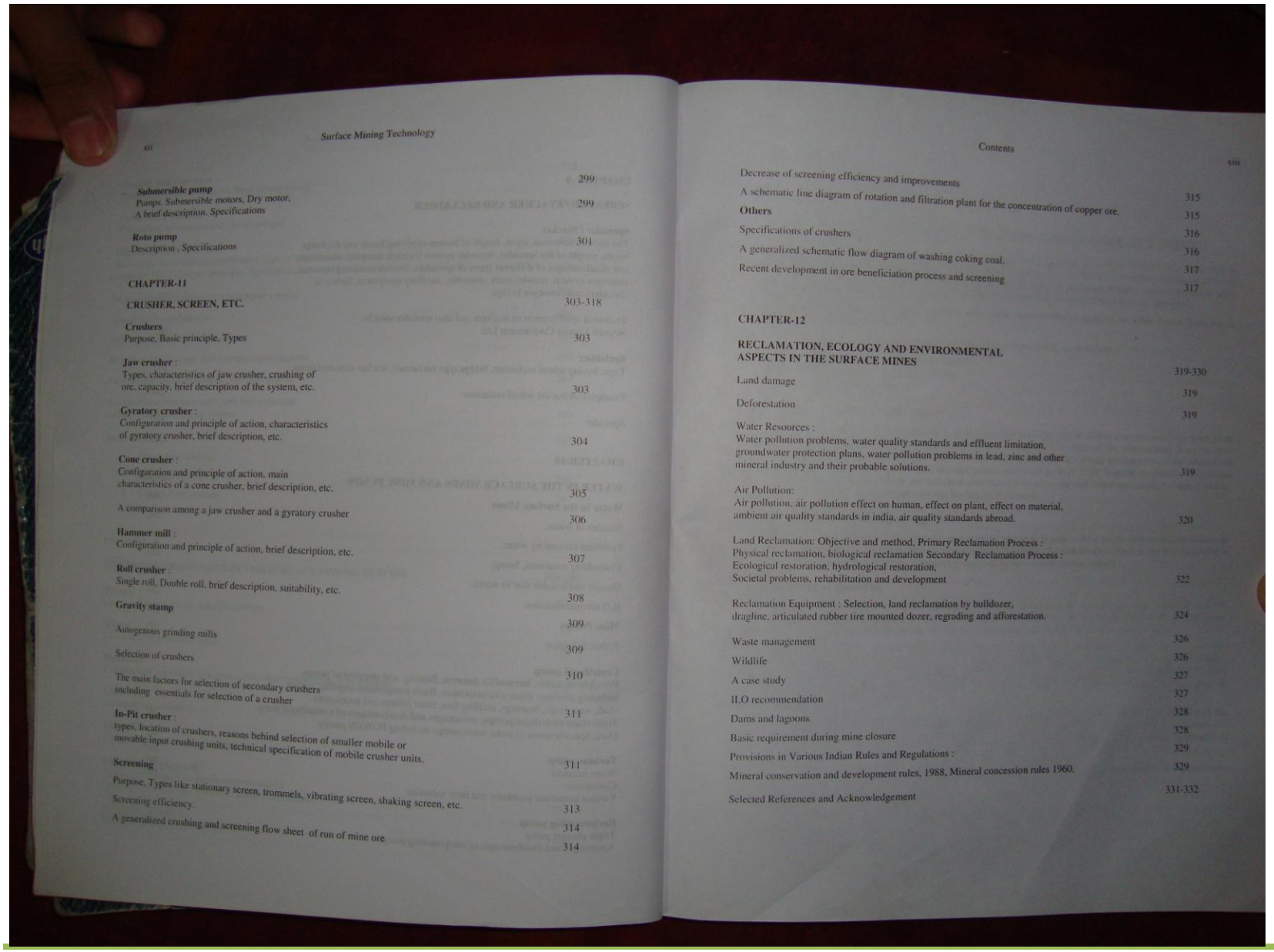

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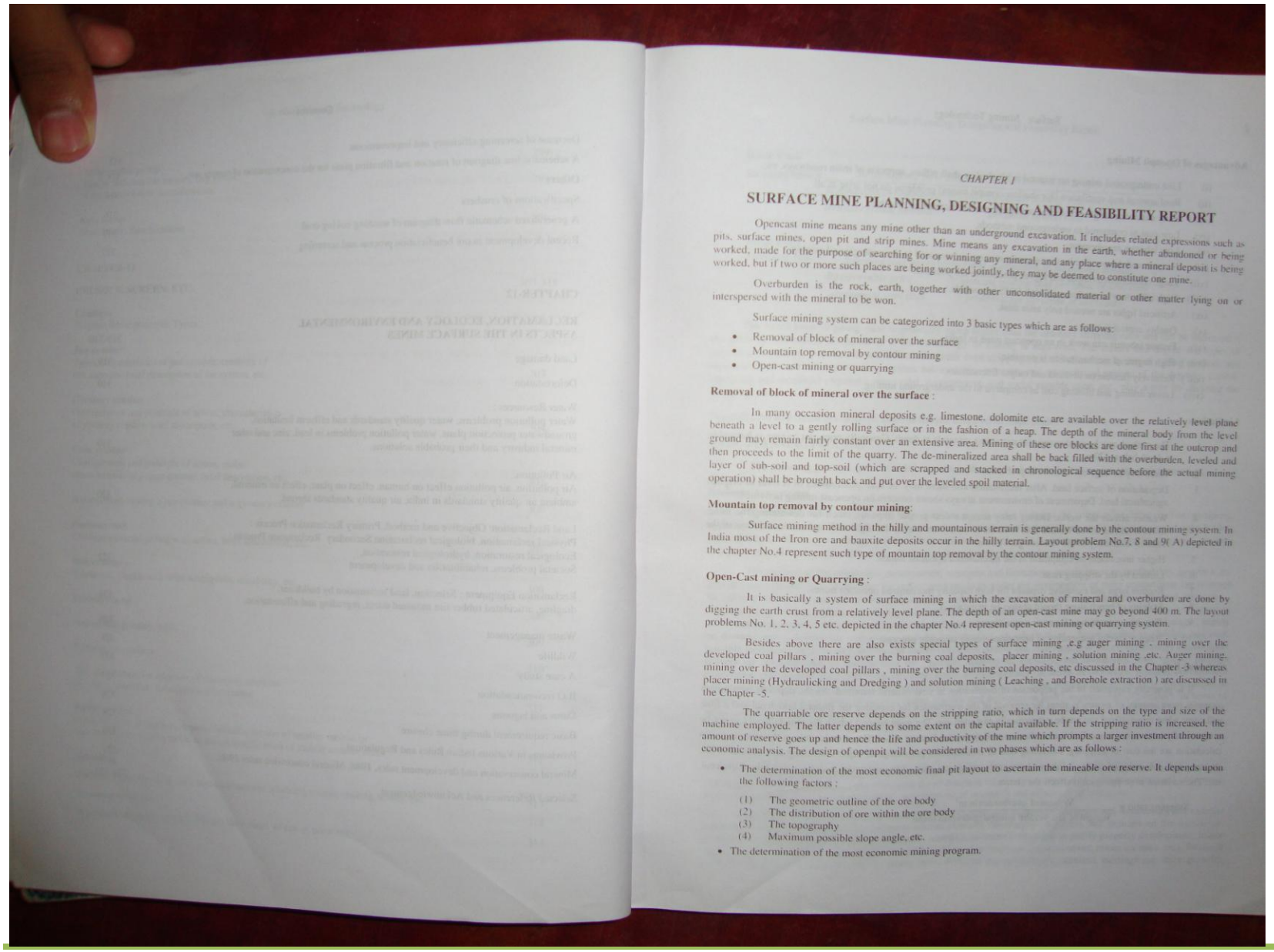


CHAPTER 1

SURFACE MINE PLANNING, DESIGNING AND FEASIBILITY REPORT

Opencast mine means any mine other than an underground excavation. It includes related expressions such as pits, surface mines, open pit and strip mines. Mine means any excavation in the earth, whether abandoned or being worked, made for the purpose of searching for or winning any mineral, and any place where a mineral deposit is being worked, but if two or more such places are being worked jointly, they may be deemed to constitute one mine.

Overburden is the rock, earth, together with other unconsolidated material or other matter lying on or interspersed with the mineral to be won.

Surface mining system can be categorized into 3 basic types which are as follows:

- Removal of block of mineral over the surface
- Mountain top removal by contour mining
- Open-cast mining or quarrying

Removal of block of mineral over the surface :

In many occasion mineral deposits e.g. limestone, dolomite etc. are available over the relatively level plane beneath a level to a gently rolling surface or in the fashion of a heap. The depth of the mineral body from the level ground may remain fairly constant over an extensive area. Mining of these ore blocks are done first at the outcrop and then proceeds to the limit of the quarry. The de-mineralized area shall be back filled with the overburden, leveled and layer of sub-soil and top-soil (which are scrapped and stacked in chronological sequence before the actual mining operation) shall be brought back and put over the leveled spoil material.

Mountain top removal by contour mining:

Surface mining method in the hilly and mountainous terrain is generally done by the contour mining system. In India most of the Iron ore and bauxite deposits occur in the hilly terrain. Layout problem No.7, 8 and 9(A) depicted in the chapter No.4 represent such type of mountain top removal by the contour mining system.

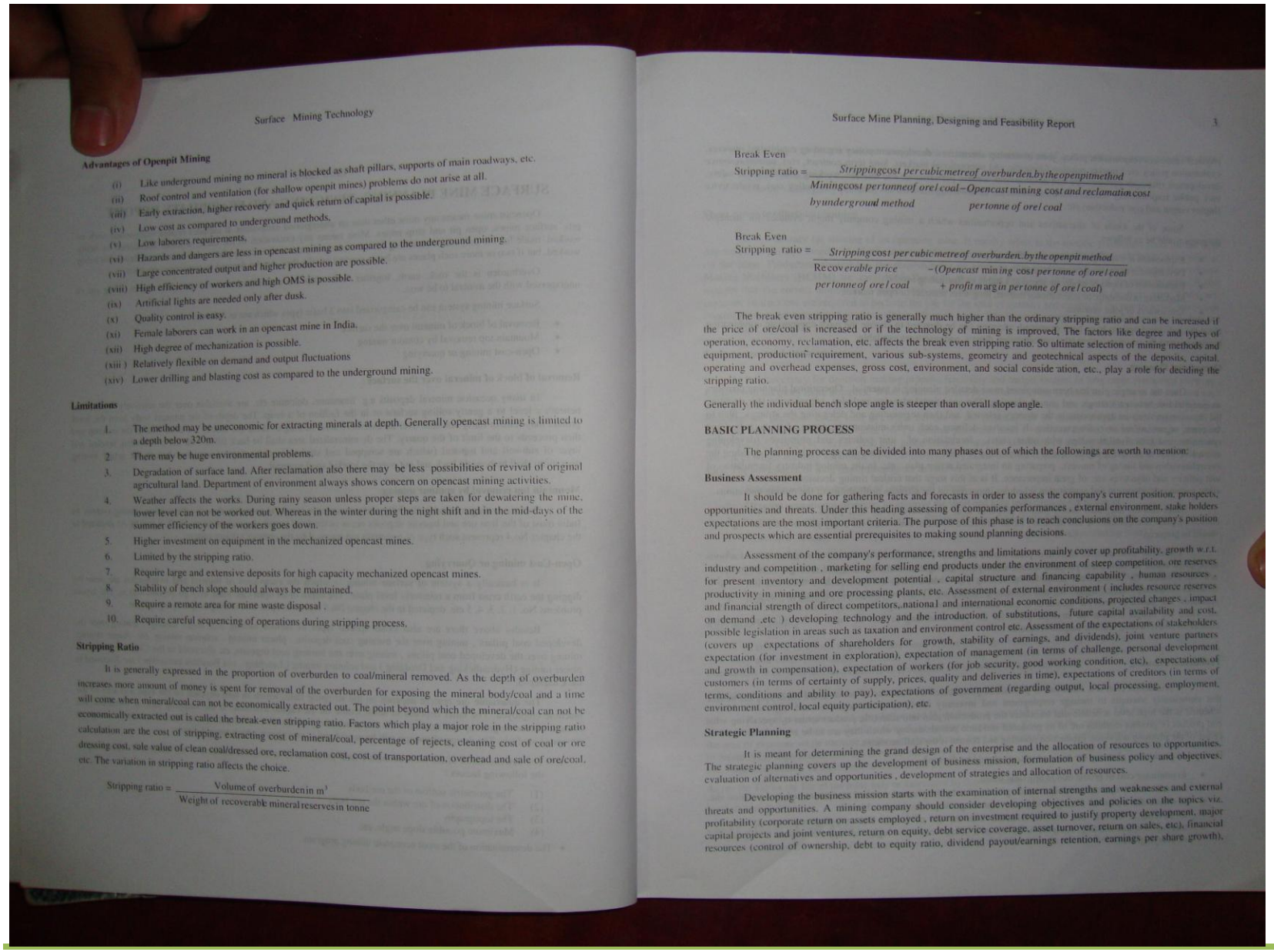
Open-Cast mining or Quarrying :

It is basically a system of surface mining in which the excavation of mineral and overburden are done by digging the earth crust from a relatively level plane. The depth of an open-cast mine may go beyond 400 m. The layout problems No. 1, 2, 3, 4, 5 etc. depicted in the chapter No.4 represent open-cast mining or quarrying system.

Besides above there are also exists special types of surface mining ,e.g auger mining , mining over the developed coal pillars , mining over the burning coal deposits, placer mining , solution mining ,etc. Auger mining, mining over the developed coal pillars , mining over the burning coal deposits, etc discussed in the Chapter -3 whereas placer mining (Hydraulicking and Dredging) and solution mining (Leaching , and Borehole extraction) are discussed in the Chapter -5.

The quarriable ore reserve depends on the stripping ratio, which in turn depends on the type and size of the machine employed. The latter depends to some extent on the capital available. If the stripping ratio is increased, the amount of reserve goes up and hence the life and productivity of the mine which prompts a larger investment through an economic analysis. The design of openpit will be considered in two phases which are as follows :

- The determination of the most economic final pit layout to ascertain the mineable ore reserve. It depends upon the following factors :
  (1) The geometric outline of the ore body
  (2) The distribution of ore within the ore body
  (3) The topography
  (4) Maximum possible slope angle, etc.
- The determination of the most economic mining program.



Advantages of Openpit Mining

(i) Like underground mining no mineral is blocked as shaft pillars, supports of main roadways, etc.
(ii) Roof control and ventilation (for shallow openpit mines) problems do not arise at all.
(iii) Early extraction, higher recovery and quick return of capital is possible.
(iv) Low cost as compared to underground methods.
(v) Low laborers requirements.
(vi) Hazards and dangers are less in opencast mining as compared to the underground mining.
(vii) Large concentrated output and higher production are possible.
(viii) High efficiency of workers and high OMS is possible.
(ix) Artificial lights are needed only after dusk.
(x) Quality control is easy.
(xi) Female laborers can work in an opencast mine in India.
(xii) High degree of mechanization is possible.
(xiii) Relatively flexible on demand and output fluctuations
(xiv) Lower drilling and blasting cost as compared to the underground mining.

Limitations

1. The method may be uneconomic for extracting minerals at depth. Generally opencast mining is limited to a depth below 320m.
2. There may be huge environmental problems.
3. Degradation of surface land. After reclamation also there may be less possibilities of revival of original agricultural land. Department of environment always shows concern on opencast mining activities.
4. Weather affects the works. During rainy season unless proper steps are taken for dewatering the mine, lower level can not be worked out. Whereas in the winter during the night shift and in the mid-days of the summer efficiency of the workers goes down.
5. Higher investment on equipment in the mechanized opencast mines.
6. Limited by the stripping ratio.
7. Require large and extensive deposits for high capacity mechanized opencast mines.
8. Stability of bench slope should always be maintained.
9. Require a remote area for mine waste disposal .
10. Require careful sequencing of operations during stripping process.

Stripping Ratio

It is generally expressed in the proportion of overburden to coal/mineral removed. As the depth of overburden increases more amount of money is spent for removal of the overburden for exposing the mineral body/coal and a time will come when mineral/coal can not be economically extracted out. The point beyond which the mineral/coal can not be economically extracted out is called the break-even stripping ratio. Factors which play a major role in the stripping ratio calculation are the cost of stripping, extracting cost of mineral/coal, percentage of rejects, cleaning cost of coal or ore dressing cost, sale value of clean coal/dressed ore, reclamation cost, cost of transportation, overhead and sale of ore/coal, etc. The variation in stripping ratio affects the choice.

$$\text{Stripping ratio} = \frac{\text{Volume of overburden in m}^3}{\text{Weight of recoverable mineral reserves in tonne}}$$

$$\text{Break Even Stripping ratio} = \frac{\text{Stripping cost per cubic metre of overburden by the openpit method}}{\text{Mining cost per tonne of ore/coal by underground method} - \text{Opencast mining cost and reclamation cost per tonne of ore/coal}}$$

$$\text{Break Even Stripping ratio} = \frac{\text{Stripping cost per cubic metre of overburden by the openpit method}}{\text{Recoverable price per tonne of ore/coal} - (\text{Opencast mining cost per tonne of ore/coal} + \text{profit margin per tonne of ore/coal})}$$

The break even stripping ratio is generally much higher than the ordinary stripping ratio and can be increased if the price of ore/coal is increased or if the technology of mining is improved. The factors like degree and types of operation, economy, reclamation, etc. affects the break even stripping ratio. So ultimate selection of mining methods and equipment, production requirement, various sub-systems, geometry and geotechnical aspects of the deposits, capital, operating and overhead expenses, gross cost, environment, and social conside ation, etc., play a role for deciding the stripping ratio.

Generally the individual bench slope angle is steeper than overall slope angle.

BASIC PLANNING PROCESS

The planning process can be divided into many phases out of which the followings are worth to mention:

Business Assessment

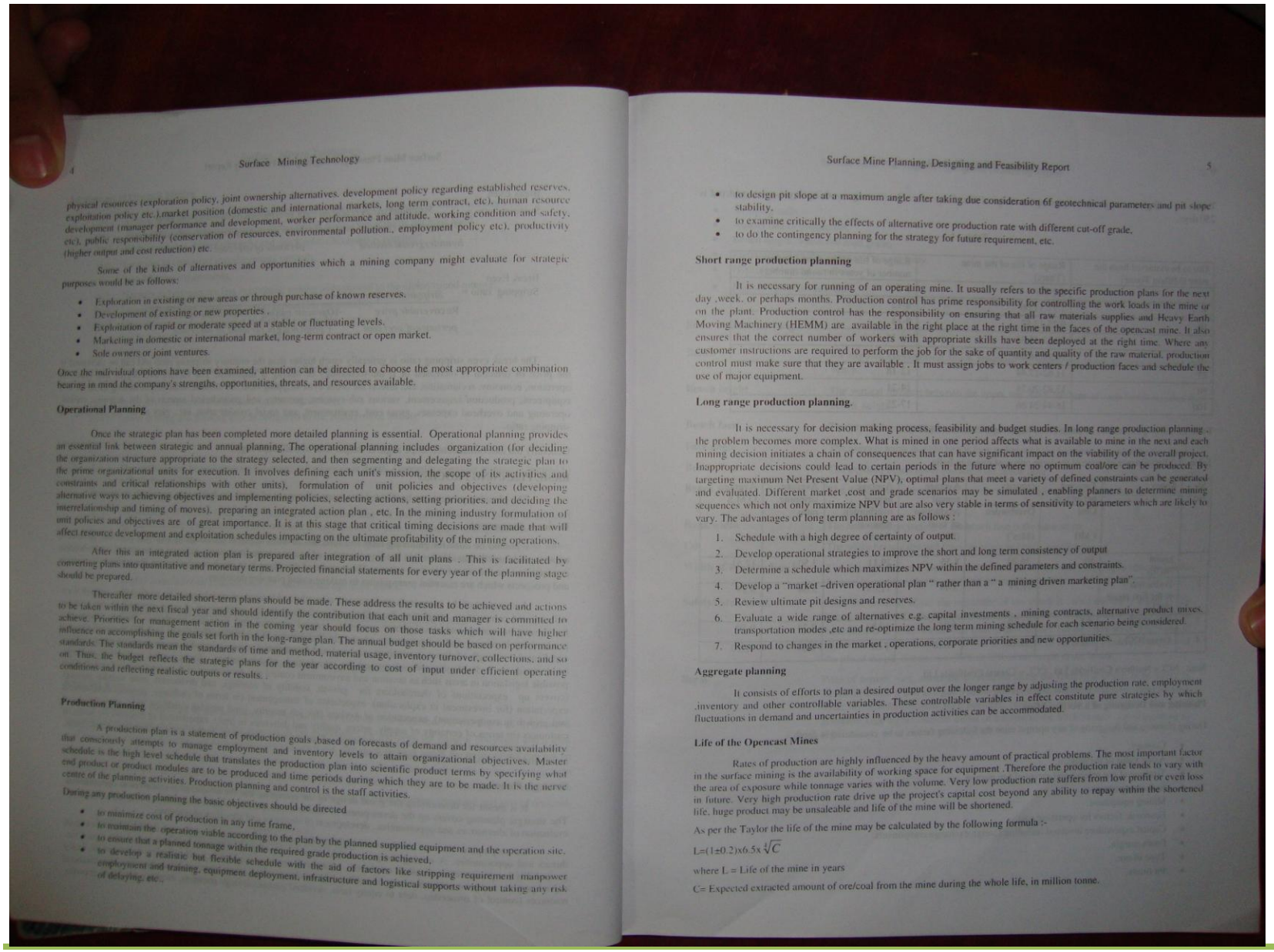
It should be done for gathering facts and forecasts in order to assess the company's current position, prospects, opportunities and threats. Under this heading assessing of companies performances , external environment, stake holders expectations are the most important criteria. The purpose of this phase is to reach conclusions on the company's position and prospects which are essential prerequisites to making sound planning decisions.

Assessment of the company's performance, strengths and limitations mainly cover up profitability, growth w.r.t. industry and competition , marketing for selling end products under the environment of steep competition, ore reserves for present inventory and development potential , capital structure and financing capability , human resources , productivity in mining and ore processing plants, etc. Assessment of external environment (includes resource reserves and financial strength of direct competitors, national and international economic conditions, projected changes , impact on demand ,etc) developing technology and the introduction of substitutions, future capital availability and cost, possible legislation in areas such as taxation and environment control etc. Assessment of the expectations of stakeholders (covers up expectations of shareholders for growth, stability of earnings, and dividends), joint venture partners expectation (for investment in exploration), expectation of management (in terms of challenge, personal development and growth in compensation), expectation of workers (for job security, good working condition, etc), expectations of customers (in terms of certainty of supply, prices, quality and deliveries in time), expectations of creditors (in terms of terms, conditions and ability to pay), expectations of government (regarding output, local processing, employment, environment control, local equity participation), etc.

Strategic Planning

It is meant for determining the grand design of the enterprise and the allocation of resources to opportunities. The strategic planning covers up the development of business mission, formulation of business policy and objectives, evaluation of alternatives and opportunities , development of strategies and allocation of resources.

Developing the business mission starts with the examination of internal strengths and weaknesses and external threats and opportunities. A mining company should consider developing objectives and policies on the topics viz. profitability (corporate return on assets employed , return on investment required to justify property development, major capital projects and joint ventures, return on equity, debt service coverage, asset turnover, return on sales, etc), financial resources (control of ownership, debt to equity ratio, dividend payout/earnings retention, earnings per share growth),



physical resources (exploration policy, joint ownership alternatives, development policy regarding established reserves, exploitation policy etc.),market position (domestic and international markets, long term contract, etc), human resource development (manager performance and development, worker performance and attitude, working condition and safety, etc), public responsibility (conservation of resources, environmental pollution., employment policy etc), productivity (higher output and cost reduction) etc.

Some of the kinds of alternatives and opportunities which a mining company might evaluate for strategic purposes would be as follows:

- Exploration in existing or new areas or through purchase of known reserves.
- Development of existing or new properties .
- Exploitation of rapid or moderate speed at a stable or fluctuating levels.
- Marketing in domestic or international market, long-term contract or open market.
- Sole owners or joint ventures.

Once the individual options have been examined, attention can be directed to choose the most appropriate combination bearing in mind the company's strengths, opportunities, threats, and resources available.

Operational Planning

Once the strategic plan has been completed more detailed planning is essential. Operational planning provides an essential link between strategic and annual planning. The operational planning includes organization (for deciding the organization structure appropriate to the strategy selected, and then segmenting and delegating the strategic plan to the prime organizational units for execution. It involves defining each unit's mission, the scope of its activities and constraints and critical relationships with other units), formulation of unit policies and objectives (developing alternative ways to achieving objectives and implementing policies, selecting actions, setting priorities, and deciding the interrelationship and timing of moves), preparing an integrated action plan , etc. In the mining industry formulation of unit policies and objectives are of great importance. It is at this stage that critical timing decisions are made that will affect resource development and exploitation schedules impacting on the ultimate profitability of the mining operations.

After this an integrated action plan is prepared after integration of all unit plans . This is facilitated by converting plans into quantitative and monetary terms. Projected financial statements for every year of the planning stage should be prepared.

Thereafter more detailed short-term plans should be made. These address the results to be achieved and actions to be taken within the next fiscal year and should identify the contribution that each unit and manager is committed to achieve. Priorities for management action in the coming year should focus on those tasks which will have higher influence on accomplishing the goals set forth in the long-range plan. The annual budget should be based on performance standards. The standards mean the standards of time and method, material usage, inventory turnover, collections, and so on. Thus, the budget reflects the strategic plans for the year according to cost of input under efficient operating conditions and reflecting realistic outputs or results. .

Production Planning

A production plan is a statement of production goals ,based on forecasts of demand and resources availability that consciously attempts to manage employment and inventory levels to attain organizational objectives. Master schedule is the high level schedule that translates the production plan into scientific product terms by specifying what end product or product modules are to be produced and time periods during which they are to be made. It is the nerve centre of the planning activities. Production planning and control is the staff activities.

During any production planning the basic objectives should be directed

- to minimize cost of production in any time frame,
- to maintain the operation viable according to the plan by the planned supplied equipment and the operation site.
- to ensure that a planned tonnage within the required grade production is achieved,
- to develop a realistic but flexible schedule with the aid of factors like stripping requirement manpower employment and training, equipment deployment, infrastructure and logistical supports without taking any risk of delaying, etc.,

- to design pit slope at a maximum angle after taking due consideration 6f geotechnical parameters and pit slope stability.
- to examine critically the effects of alternative ore production rate with different cut-off grade.
- to do the contingency planning for the strategy for future requirement, etc.

Short range production planning

It is necessary for running of an operating mine. It usually refers to the specific production plans for the next day ,week, or perhaps months. Production control has prime responsibility for controlling the work loads in the mine or on the plant. Production control has the responsibility on ensuring that all raw materials supplies and Heavy Earth Moving Machinery (HEMM) are available in the right place at the right time in the faces of the opencast mine. It also ensures that the correct number of workers with appropriate skills have been deployed at the right time. Where any customer instructions are required to perform the job for the sake of quantity and quality of the raw material, production control must make sure that they are available . It must assign jobs to work centers / production faces and schedule the use of major equipment.

Long range production planning.

It is necessary for decision making process, feasibility and budget studies. In long range production planning , the problem becomes more complex. What is mined in one period affects what is available to mine in the next and each mining decision initiates a chain of consequences that can have significant impact on the viability of the overall project. Inappropriate decisions could lead to certain periods in the future where no optimum coal/ore can be produced. By targeting maximum Net Present Value (NPV), optimal plans that meet a variety of defined constraints can be generated and evaluated. Different market ,cost and grade scenarios may be simulated , enabling planners to determine mining sequences which not only maximize NPV but are also very stable in terms of sensitivity to parameters which are likely to vary. The advantages of long term planning are as follows :

1. Schedule with a high degree of certainty of output.
2. Develop operational strategies to improve the short and long term consistency of output
3. Determine a schedule which maximizes NPV within the defined parameters and constraints.
4. Develop a "market –driven operational plan " rather than a " a mining driven marketing plan".
5. Review ultimate pit designs and reserves.
6. Evaluate a wide range of alternatives e.g. capital investments , mining contracts, alternative product mixes, transportation modes ,etc and re-optimize the long term mining schedule for each scenario being considered.
7. Respond to changes in the market , operations, corporate priorities and new opportunities.

Aggregate planning

It consists of efforts to plan a desired output over the longer range by adjusting the production rate, employment ,inventory and other controllable variables. These controllable variables in effect constitute pure strategies by which fluctuations in demand and uncertainties in production activities can be accommodated.

Life of the Opencast Mines

Rates of production are highly influenced by the heavy amount of practical problems. The most important factor in the surface mining is the availability of working space for equipment .Therefore the production rate tends to vary with the area of exposure while tonnage varies with the volume. Very low production rate suffers from low profit or even loss in future. Very high production rate drive up the project's capital cost beyond any ability to repay within the shortened life. huge product may be unsaleable and life of the mine will be shortened.

As per the Taylor the life of the mine may be calculated by the following formula :-

$L=(1\pm0.2)\times6.5\times\sqrt[4]{C}$

where L = Life of the mine in years

C= Expected extracted amount of ore/coal from the mine during the whole life, in million tonne.

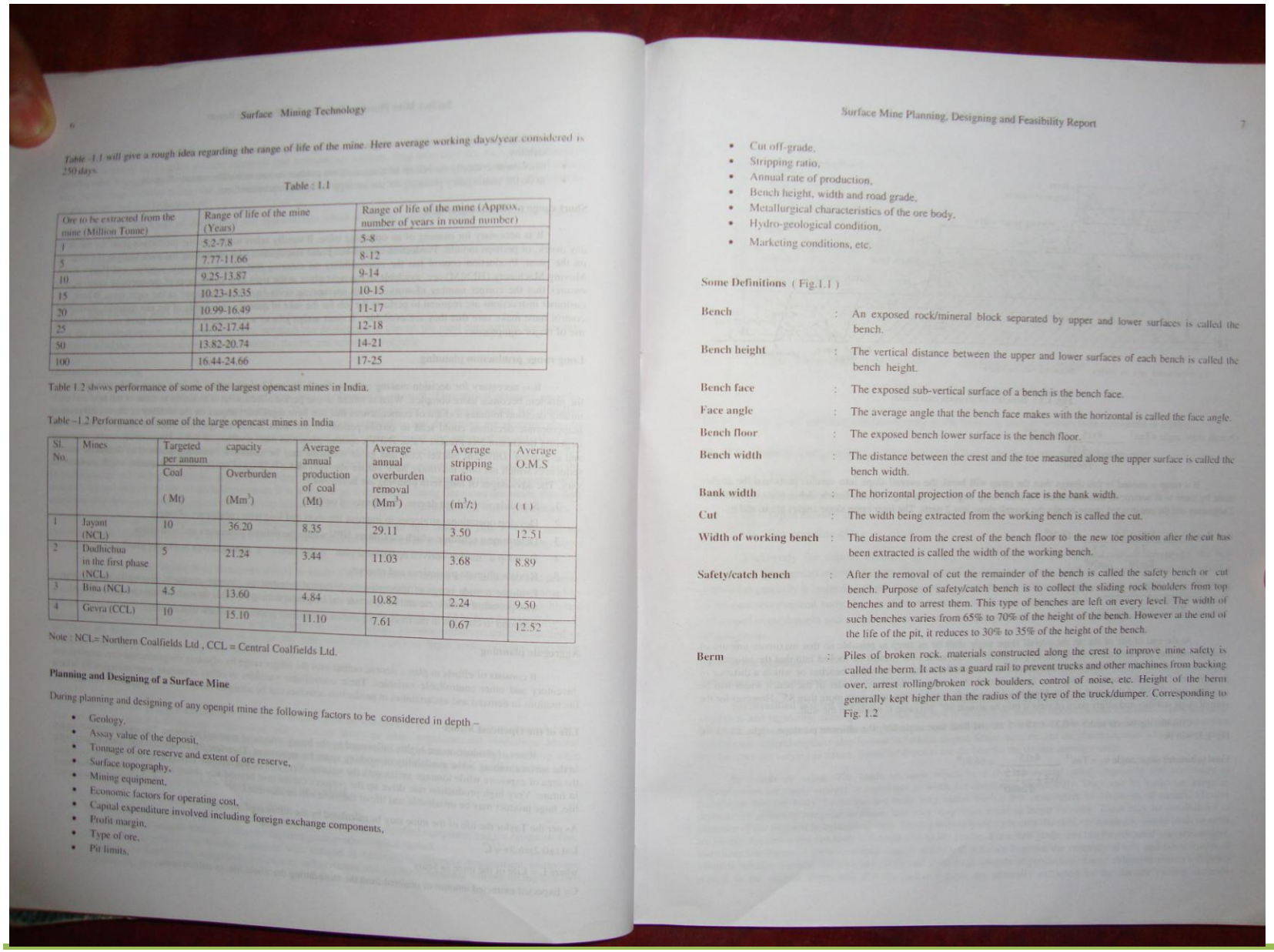


Table –1.1 will give a rough idea regarding the range of life of the mine. Here average working days/year considered is 250 days.

Table : 1.1

Ore to be extracted from the mine (Million Tonne)	Range of life of the mine (Years)	Range of life of the mine (Approx. number of years in round number)
1	5.2-7.8	5-8
5	7.77-11.66	8-12
10	9.25-13.87	9-14
15	10.23-15.35	10-15
20	10.99-16.49	11-17
25	11.62-17.44	12-18
50	13.82-20.74	14-21
100	16.44-24.66	17-25

Table 1.2 shows performance of some of the largest opencast mines in India.

Table –1.2 Performance of some of the large opencast mines in India

Sl. No.	Mines	Targeted capacity per annum		Average annual production of coal (Mt)	Average annual overburden removal (Mm^3)	Average stripping ratio (m^3/t)	Average O.M.S (t)
		Coal (Mt)	Overburden (Mm^3)				
1	Jayant (NCL)	10	36.20	8.35	29.11	3.50	12.51
2	Dudhichua in the first phase (NCL)	5	21.24	3.44	11.03	3.68	8.89
3	Bina (NCL)	4.5	13.60	4.84	10.82	2.24	9.50
4	Gevra (CCL)	10	15.10	11.10	7.61	0.67	12.52

Note : NCL= Northern Coalfields Ltd , CCL = Central Coalfields Ltd.

Planning and Designing of a Surface Mine

During planning and designing of any openpit mine the following factors to be considered in depth –

- Geology,
- Assay value of the deposit,
- Tonnage of ore reserve and extent of ore reserve,
- Surface topography,
- Mining equipment,
- Economic factors for operating cost,
- Capital expenditure involved including foreign exchange components,
- Profit margin,
- Type of ore,
- Pit limits,

- Cut off-grade,
- Stripping ratio,
- Annual rate of production,
- Bench height, width and road grade,
- Metallurgical characteristics of the ore body,
- Hydro-geological condition,
- Marketing conditions, etc.

Some Definitions (Fig.1.1)

Bench : An exposed rock/mineral block separated by upper and lower surfaces is called the bench.

Bench height : The vertical distance between the upper and lower surfaces of each bench is called the bench height.

Bench face : The exposed sub-vertical surface of a bench is the bench face.

Face angle : The average angle that the bench face makes with the horizontal is called the face angle.

Bench floor : The exposed bench lower surface is the bench floor.

Bench width : The distance between the crest and the toe measured along the upper surface is called the bench width.

Bank width : The horizontal projection of the bench face is the bank width.

Cut : The width being extracted from the working bench is called the cut.

Width of working bench : The distance from the crest of the bench floor to the new toe position after the cut has been extracted is called the width of the working bench.

Safety/catch bench : After the removal of cut the remainder of the bench is called the safety bench or cut bench. Purpose of safety/catch bench is to collect the sliding rock boulders from top benches and to arrest them. This type of benches are left on every level. The width of such benches varies from 65% to 70% of the height of the bench. However at the end of the life of the pit, it reduces to 30% to 35% of the height of the bench.

Berm : Piles of broken rock, materials constructed along the crest to improve mine safety is called the berm. It acts as a guard rail to prevent trucks and other machines from backing over, arrest rolling/broken rock boulders, control of noise, etc. Height of the berm generally kept higher than the radius of the tyre of the truck/dumper. Corresponding to Fig. 1.2

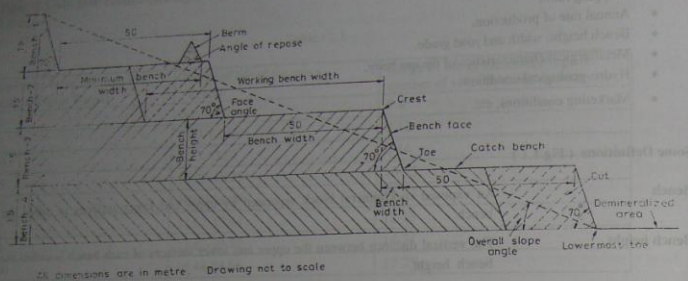


Fig. 1.1 : Benches of an open cast mines

$$\text{Overall slope angle} = \text{Tan}^{-1}\frac{4x15}{3x50+\frac{4x15}{Tan70^0}} = 19.23^0$$

If a ramp is needed in this design then the ramp will break the overall slope into various parts and the angles made by them with horizontal axis are called the inter ramp angles. In the Fig. 1.2 ,the bench -3 has been divided into 2 segments and the constructed ramps breaks the overall slope into 2 parts. The inter ramp slope angles are α_1 and α_2.

$$\alpha_1 = \text{Tan}^{-1}\frac{2x15+7.5}{2x50+\frac{7.5}{Tan70^0}+\frac{7.5}{Tan70^0}} = 18.26^0$$

$$\alpha_2 = \text{Tan}^{-1}\frac{7.5+15}{50+\frac{7.5}{Tan70^0}+\frac{15}{Tan70^0}} = 21.14^0$$

At the end of life of the pit the overall slope angle shall be as steep as possible so that maximum amount of mineral/coal may be extracted from the developed block. At the same time it must be looked into that the ultimate pit slope should be stable. The bench shall be reduced to less than the width of the safety benches or within a distance of 5 m. Some of the safety benches may even be eliminated. As a rule of thumb remainder of the bench width will be approximately 0.33% of the bench height. Final bench face angle may be very steep even more than 85°. However for the sake of slope stability and safety point of view it may be around 80°. No ramp is needed in the wall.

Considering bench width = 0.33 x 15 = 5 m, and final face angle 80°, the ultimate pit slope angle (w.r.t the Fig-1.3) will be-

$$\text{Final (ultimate) slope angle } \alpha_3 = \text{Tan}^{-1}\frac{4x15}{3x5+\frac{4x15}{Tan80^0}} = 66.91^0$$

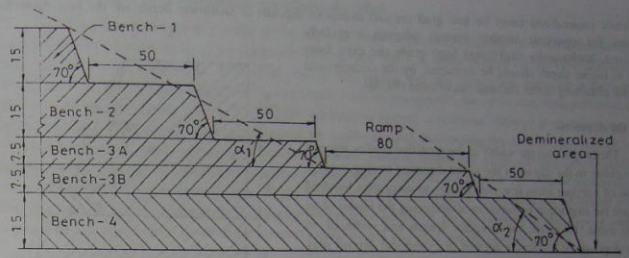


Fig. 1.2 : Interramp constructed in the Bench - 3

During designing and planning a surface mining pit the following main factors are to be considered in depth :

Dip of the Deposit- Categorically it can be divided into 4 ways in flat terrain and stratified deposits:

(a) Dip below 7°

(b) Dip in between 7° to 16°

(c) Dip in between 16° to 45°

(d) Dip more than 45°

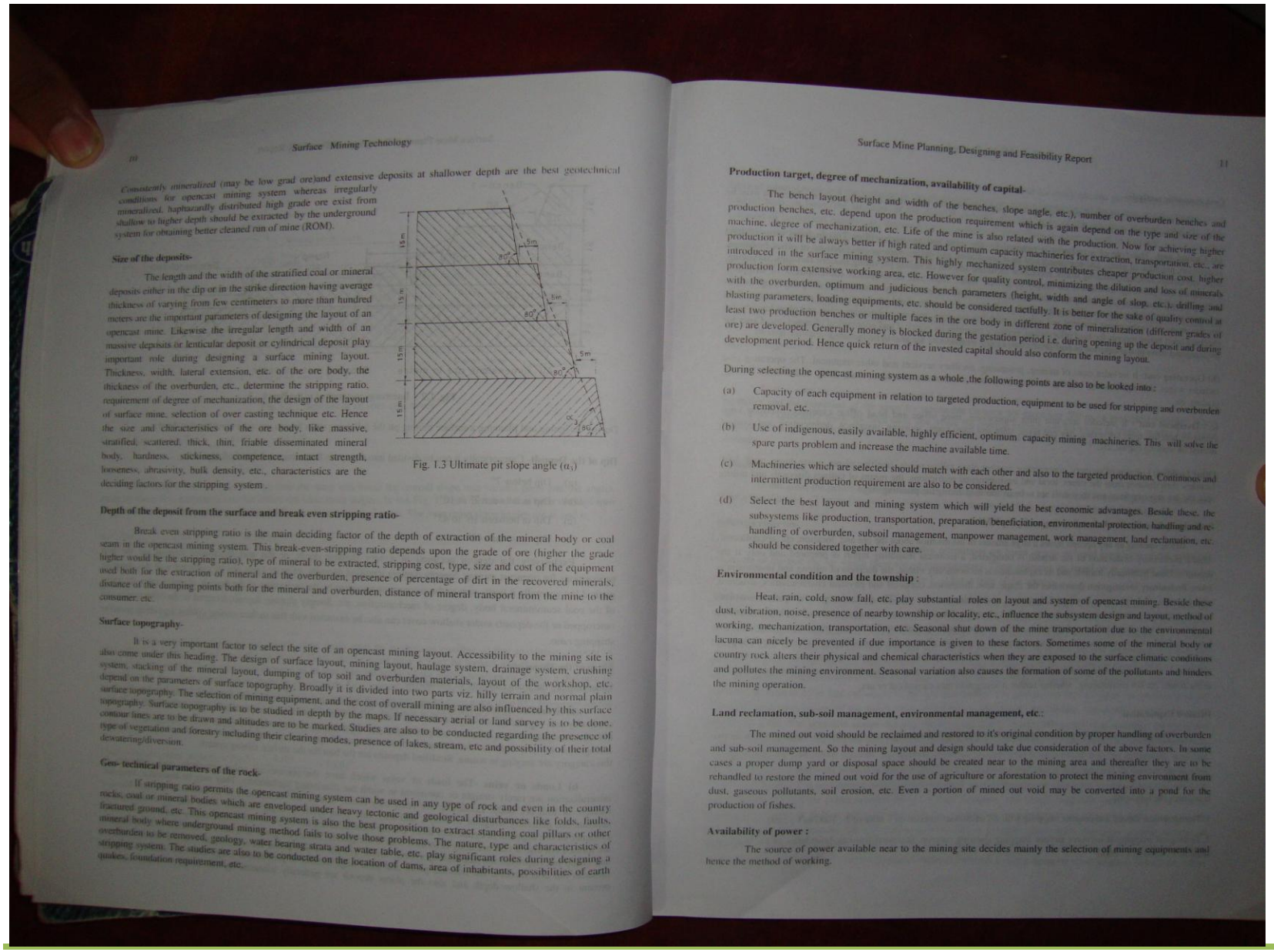
Exclusively flat deposit occurred in a very shallow depth, or flat and outcropped deposit are the best propositions for opencast mining. Steeply dipping stratified deposits, veins or loads can be extracted with this opencast method but generally it limits near to the outcrops and governs by the inclination of the deposit, stripping ratio, thickness of the coal seam/mineral body, degree of mechanization, etc. Steeply dipping deposit occurred in the hilly terrain and outcropped or the deposits under shallow covet can also be extracted with this surface mining system depending upon the stripping ratio.

Form of the Deposits-

Following are the main category of formations :

(a) Stratified bedded deposits -These types of deposits are mostly sedimentary deposits having a high degree of continuity and regularity, although sometimes the bed is affected by various discontinuities like folds, faults and other disturbances. Coal seams, lime stone deposits, iron ore deposits, etc. come under this category. Sometimes metamorphic and igneous mineral deposits also formed in bedded shapes. Shape, size, etc., of the placer deposits which comes under this category are varying in nature. Stratified deposits are the best for the surface mining system.

b) Loads or veins -The loads or veins which have the commercial value, regularity and continuity of mineralization are rarely straight or consistent in width but distributed haphazardly. They may be lenticular, irregular, massive or thin ore body. Lenticular ore body may extent from a few centimeters to thousands of meters. Massive deposits are generally irregular in shape, and may vary from small to larger in sizes. There may be possibilities of existence of ore deposits in cylindrical form. During mineralization not only the rich, economically viable loads or veins are formed but also a vast ore body is formed with lower grades. These low grade ores can be extracted with the modern and cheap but improved surface mining system which will not only improve the recovery of ore, and conservation of minerals but also increase the ultimate profit and yield huge amount of production. Large extensive massive deposits present in the shallow depth and also the placer deposit are generally extracted by the surface mining methods.



10 Surface Mining Technology

Consistently mineralized (may be low grad ore)and extensive deposits at shallower depth are the best geotechnical conditions for opencast mining system whereas irregularly mineralized, haphazardly distributed high grade ore exist from shallow to higher depth should be extracted by the underground system for obtaining better cleaned run of mine (ROM).

Size of the deposits-

The length and the width of the stratified coal or mineral deposits either in the dip or in the strike direction having average thickness of varying from few centimeters to more than hundred meters are the important parameters of designing the layout of an opencast mine. Likewise the irregular length and width of an massive deposits or lenticular deposit or cylindrical deposit play important role during designing a surface mining layout. Thickness, width, lateral extension, etc. of the ore body, the thickness of the overburden, etc., determine the stripping ratio, requirement of degree of mechanization, the design of the layout of surface mine, selection of over casting technique etc. Hence the size and characteristics of the ore body, like massive, stratified, scattered, thick, thin, friable disseminated mineral body, hardness, stickiness, competence, intact strength, looseness, abrasivity, bulk density, etc., characteristics are the deciding factors for the stripping system.

Fig. 1.3 Ultimate pit slope angle (α_s)

Depth of the deposit from the surface and break even stripping ratio-

Break even stripping ratio is the main deciding factor of the depth of extraction of the mineral body or coal seam in the opencast mining system. This break-even-stripping ratio depends upon the grade of ore (higher the grade higher would be the stripping ratio), type of mineral to be extracted, stripping cost, type, size and cost of the equipment used both for the extraction of mineral and the overburden, presence of percentage of dirt in the recovered minerals, distance of the dumping points both for the mineral and overburden, distance of mineral transport from the mine to the consumer etc.

Surface topography-

It is a very important factor to select the site of an opencast mining layout. Accessibility to the mining site is also come under this heading. The design of surface layout, mining layout, haulage system, drainage system, crushing system, stacking of the mineral layout, dumping of top soil and overburden materials, layout of the workshop, etc. depend on the parameters of surface topography. Broadly it is divided into two parts viz. hilly terrain and normal plain surface topography. The selection of mining equipment, and the cost of overall mining are also influenced by this surface topography. Surface topography is to be studied in depth by the maps. If necessary aerial or land survey is to be done, contour lines are to be drawn and altitudes are to be marked. Studies are also to be conducted regarding the presence of type of vegetation and forestry including their clearing modes, presence of lakes, stream, etc and possibility of their total dewatering/diversion.

Geo- technical parameters of the rock-

If stripping ratio permits the opencast mining system can be used in any type of rock and even in the country rocks, coal or mineral bodies which are enveloped under heavy tectonic and geological disturbances like folds, faults, fractured ground, etc. This opencast mining system is also the best proposition to extract standing coal pillars or other mineral body where underground mining method fails to solve those problems. The nature, type and characteristics of overburden to be removed, geology, water bearing strata and water table, etc. play significant roles during designing a stripping system. The studies are also to be conducted on the location of dams, area of inhabitants, possibilities of earth quakes, foundation requirement, etc.

Surface Mine Planning, Designing and Feasibility Report 11

Production target, degree of mechanization, availability of capital-

The bench layout (height and width of the benches, slope angle, etc.), number of overburden benches and production benches, etc. depend upon the production requirement which is again depend on the type and size of the machine, degree of mechanization, etc. Life of the mine is also related with the production. Now for achieving higher production it will be always better if high rated and optimum capacity machineries for extraction, transportation, etc., are introduced in the surface mining system. This highly mechanized system contributes cheaper production cost, higher production form extensive working area, etc. However for quality control, minimizing the dilution and loss of minerals with the overburden, optimum and judicious bench parameters (height, width and angle of slop, etc.), drilling and blasting parameters, loading equipments, etc. should be considered tactfully. It is better for the sake of quality control at least two production benches or multiple faces in the ore body in different zone of mineralization (different grades of ore) are developed. Generally money is blocked during the gestation period i.e. during opening up the deposit and during development period. Hence quick return of the invested capital should also conform the mining layout.

During selecting the opencast mining system as a whole ,the following points are also to be looked into :

(a) Capacity of each equipment in relation to targeted production, equipment to be used for stripping and overburden removal, etc.

(b) Use of indigenous, easily available, highly efficient, optimum capacity mining machineries. This will solve the spare parts problem and increase the machine available time.

(c) Machineries which are selected should match with each other and also to the targeted production. Continuous and intermittent production requirement are also to be considered.

(d) Select the best layout and mining system which will yield the best economic advantages. Beside these, the subsystems like production, transportation, preparation, beneficiation, environmental protection, handling and re-handling of overburden, subsoil management, manpower management, work management, land reclamation, etc. should be considered together with care.

Environmental condition and the township :

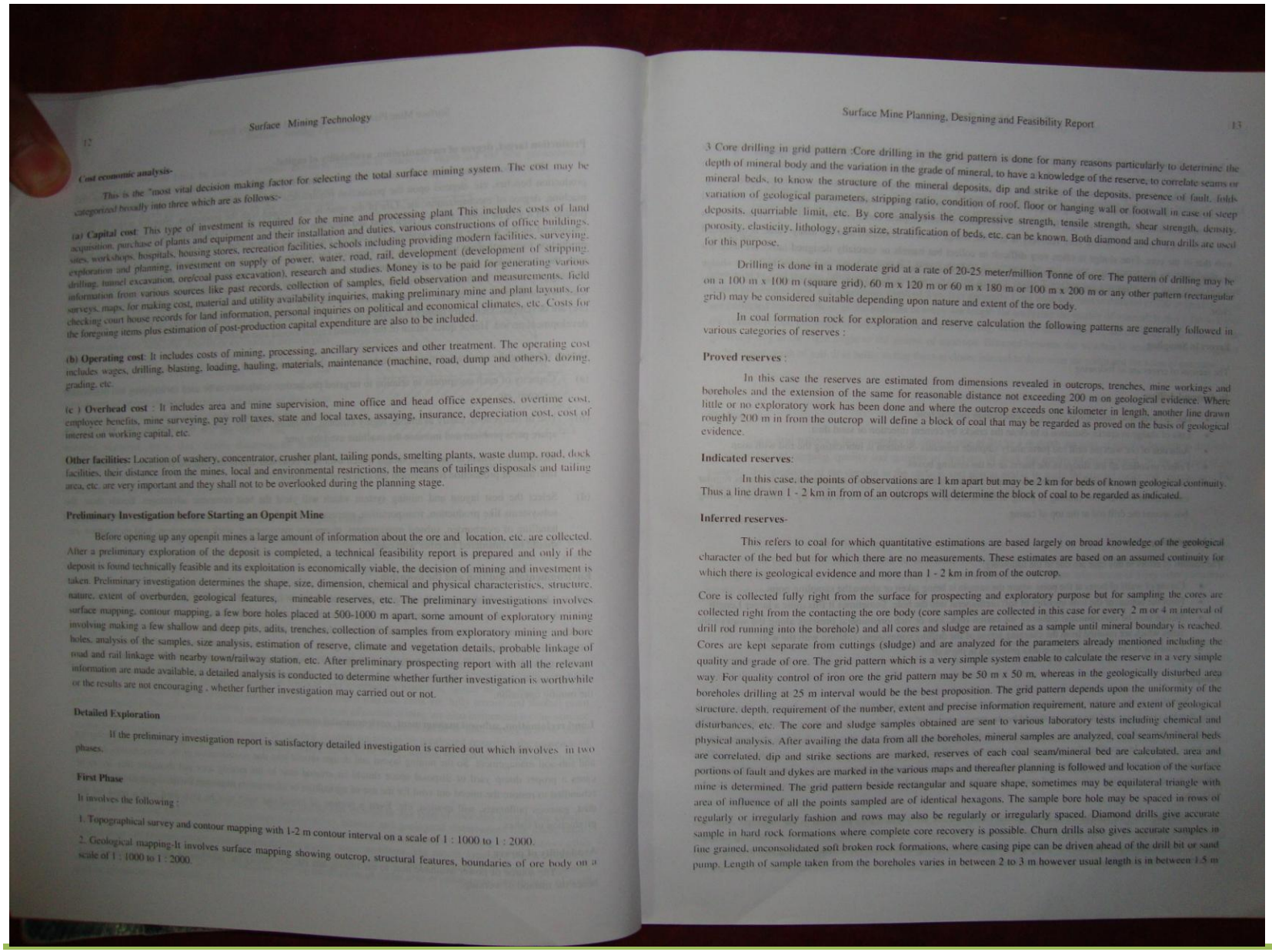
Heat, rain, cold, snow fall, etc. play substantial roles on layout and system of opencast mining. Beside these dust, vibration, noise, presence of nearby township or locality, etc., influence the subsystem design and layout, method of working, mechanization, transportation, etc. Seasonal shut down of the mine transportation due to the environmental lacuna can nicely be prevented if due importance is given to these factors. Sometimes some of the mineral body or country rock alters their physical and chemical characteristics when they are exposed to the surface climatic conditions and pollutes the mining environment. Seasonal variation also causes the formation of some of the pollutants and hinders the mining operation.

Land reclamation, sub-soil management, environmental management, etc.:

The mined out void should be reclaimed and restored to it's original condition by proper handling of overburden and sub-soil management. So the mining layout and design should take due consideration of the above factors. In some cases a proper dump yard or disposal space should be created near to the mining area and thereafter they are to be rehandled to restore the mined out void for the use of agriculture or aforestation to protect the mining environment from dust, gaseous pollutants, soil erosion, etc. Even a portion of mined out void may be converted into a pond for the production of fishes.

Availability of power :

The source of power available near to the mining site decides mainly the selection of mining equipments and hence the method of working.



Cost economic analysis-

This is the "most vital decision making factor for selecting the total surface mining system. The cost may be categorized broadly into three which are as follows:-

(a) Capital cost: This type of investment is required for the mine and processing plant This includes costs of land acquisition, purchase of plants and equipment and their installation and duties, various constructions of office buildings, sites, workshops, hospitals, housing stores, recreation facilities, schools including providing modern facilities, surveying, exploration and planning, investment on supply of power, water, road, rail, development (development of stripping, drilling, tunnel excavation, ore/coal pass excavation), research and studies. Money is to be paid for generating various information from various sources like past records, collection of samples, field observation and measurements, field surveys, maps, for making cost, material and utility availability inquiries, making preliminary mine and plant layouts, for checking court house records for land information, personal inquiries on political and economical climates, etc. Costs for the foregoing items plus estimation of post-production capital expenditure are also to be included.

(b) Operating cost: It includes costs of mining, processing, ancillary services and other treatment. The operating cost includes wages, drilling, blasting, loading, hauling, materials, maintenance (machine, road, dump and others), dozing, grading, etc.

(c) Overhead cost : It includes area and mine supervision, mine office and head office expenses, overtime cost, employee benefits, mine surveying, pay roll taxes, state and local taxes, assaying, insurance, depreciation cost, cost of interest on working capital, etc.

Other facilities: Location of washery, concentrator, crusher plant, tailing ponds, smelting plants, waste dump, road, dock facilities, their distance from the mines, local and environmental restrictions, the means of tailings disposals and tailing area, etc. are very important and they shall not to be overlooked during the planning stage.

Preliminary Investigation before Starting an Openpit Mine

Before opening up any openpit mines a large amount of information about the ore and location, etc. are collected. After a preliminary exploration of the deposit is completed, a technical feasibility report is prepared and only if the deposit is found technically feasible and its exploitation is economically viable, the decision of mining and investment is taken. Preliminary investigation determines the shape, size, dimension, chemical and physical characteristics, structure, nature, extent of overburden, geological features, mineable reserves, etc. The preliminary investigations involves surface mapping, contour mapping, a few bore holes placed at 500-1000 m apart, some amount of exploratory mining involving making a few shallow and deep pits, adits, trenches, collection of samples from exploratory mining and bore holes, analysis of the samples, size analysis, estimation of reserve, climate and vegetation details, probable linkage of road and rail linkage with nearby town/railway station, etc. After preliminary prospecting report with all the relevant information are made available, a detailed analysis is conducted to determine whether further investigation is worthwhile or the results are not encouraging , whether further investigation may carried out or not.

Detailed Exploration

If the preliminary investigation report is satisfactory detailed investigation is carried out which involves in two phases.

First Phase

It involves the following :

1. Topographical survey and contour mapping with 1-2 m contour interval on a scale of 1 : 1000 to 1 : 2000.

2. Geological mapping-It involves surface mapping showing outcrop, structural features, boundaries of ore body on a scale of 1 : 1000 to 1 : 2000.

3 Core drilling in grid pattern :Core drilling in the grid pattern is done for many reasons particularly to determine the depth of mineral body and the variation in the grade of mineral, to have a knowledge of the reserve, to correlate seams or mineral beds, to know the structure of the mineral deposits, dip and strike of the deposits, presence of fault, folds variation of geological parameters, stripping ratio, condition of roof, floor or hanging wall or footwall in case of steep deposits, quarriable limit, etc. By core analysis the compressive strength, tensile strength, shear strength, density, porosity, elasticity, lithology, grain size, stratification of beds, etc. can be known. Both diamond and churn drills are used for this purpose.

Drilling is done in a moderate grid at a rate of 20-25 meter/million Tonne of ore. The pattern of drilling may be on a 100 m x 100 m (square grid), 60 m x 120 m or 60 m x 180 m or 100 m x 200 m or any other pattern (rectangular grid) may be considered suitable depending upon nature and extent of the ore body.

In coal formation rock for exploration and reserve calculation the following patterns are generally followed in various categories of reserves :

Proved reserves :

In this case the reserves are estimated from dimensions revealed in outcrops, trenches, mine workings and boreholes and the extension of the same for reasonable distance not exceeding 200 m on geological evidence. Where little or no exploratory work has been done and where the outcrop exceeds one kilometer in length, another line drawn roughly 200 m in from the outcrop will define a block of coal that may be regarded as proved on the basis of geological evidence.

Indicated reserves:

In this case, the points of observations are 1 km apart but may be 2 km for beds of known geological continuity. Thus a line drawn 1 - 2 km in from of an outcrops will determine the block of coal to be regarded as indicated.

Inferred reserves-

This refers to coal for which quantitative estimations are based largely on broad knowledge of the geological character of the bed but for which there are no measurements. These estimates are based on an assumed continuity for which there is geological evidence and more than 1 - 2 km in from of the outcrop.

Core is collected fully right from the surface for prospecting and exploratory purpose but for sampling the cores are collected right from the contacting the ore body (core samples are collected in this case for every 2 m or 4 m interval of drill rod running into the borehole) and all cores and sludge are retained as a sample until mineral boundary is reached. Cores are kept separate from cuttings (sludge) and are analyzed for the parameters already mentioned including the quality and grade of ore. The grid pattern which is a very simple system enable to calculate the reserve in a very simple way. For quality control of iron ore the grid pattern may be 50 m x 50 m, whereas in the geologically disturbed area boreholes drilling at 25 m interval would be the best proposition. The grid pattern depends upon the uniformity of the structure, depth, requirement of the number, extent and precise information requirement, nature and extent of geological disturbances, etc. The core and sludge samples obtained are sent to various laboratory tests including chemical and physical analysis. After availing the data from all the boreholes, mineral samples are analyzed, coal seams/mineral beds are correlated, dip and strike sections are marked, reserves of each coal seam/mineral bed are calculated, area and portions of fault and dykes are marked in the various maps and thereafter planning is followed and location of the surface mine is determined. The grid pattern beside rectangular and square shape, sometimes may be equilateral triangle with area of influence of all the points sampled are of identical hexagons. The sample bore hole may be spaced in rows of regularly or irregularly fashion and rows may also be regularly or irregularly spaced. Diamond drills give accurate sample in hard rock formations where complete core recovery is possible. Churn drills also gives accurate samples in fine grained, unconsolidated soft broken rock formations, where casing pipe can be driven ahead of the drill bit or sand pump. Length of sample taken from the boreholes varies in between 2 to 3 m however usual length is in between 1.5 m

to 2 m. In soft rock formation where caving is a regular features and mineralization is irregular, shorter length of samples are advisable for precise and accurate work. In firm and strong ground condition where mineralization has a uniform value, longer length will give adequate information and save expenses for sampling and assaying. In large scale boring, the calculation also simplified by taking samples in 2 m, 3 m or 4 m lengths and measurements are recorded. Diamond drill sample consists of core and sludge. In hard rock formation where complete cores are obtained they are assayed. In soft broken rock formations where core recovery is low the sludge must be collected and it's assay value is combined with that of the core. Fine sludge is often very difficult to collect but barrels or specially designed tanks are used for settling sludge. Accurate results may be obtained by assay analysis of the sludge if large quantity of barrel or sludge tanks are used, they are filled and emptied in rotation to allow entyre flow from the hole during a sample run to settle clear.

Errors in Sampling

The reasons of errors are as following :

In diamond drilling

- Due to caved hole -solution is casing or cementation into the hole.
- Loss of sludge in cracks -Solution is to close the cracks by cement injection or sand dust.
- Adhesion of ore with the drill rod particularly sulphide minerals -Solution is lubricating the rod with soap.
- Failure to collect all the sludge in the barrel or in the settling boxes.
- Failure to collect all sludge particularly coarse and heavier ore at the end of sample run--Solution is regular examination of overflow, running of pump at full speed at the reverse direction of flow by a temporary stuffing box around the drill rod at the top of casing.

In churn drilling

The reasons of errors in this case are due to the following:

- Caving of walls of holes at the point where the sample is being taken or above the sampling point.
- From the bottom of the hole inrushes of soft ore/mineral into the hole. Sometimes it fills quite a larger Length.
- Sludge may be contaminated either by the barren rock, low or rich ore/mineral presents along the wall of the holes which are resulted from the churning action of the drilling components rubbing with the wall.
- Losses of the total sludge in the cracks presented in the wall.

Sampling

Sampling is the process of obtaining a small amount/portion of ore/coal/rock substances from the mass in such away that the consistency of the portion should be the representative of the whole mass. Sampling is needed to know the value of the ore/coal and it is done by accurate examination of mining property, the physical, chemical characteristics of the ore or mineral. In sampling the 'assay value' is termed for all characteristics, physical or chemical, which may affect the value of the ore/mineral like the actual metal content in the ore of gold ,silver, etc., thermal unit, fixed carbon, volatile matter, ash, caking index, etc. of coal, in iron the complete analysis of the ore for various constituents like Fe, SiO_2, etc.

Sampling may be conducted in any exposed surface of ore body which may be found in the outcrops, adits, etc. Because of weathering in the exposed surface trenches, adits, drifts, deep pits, etc. may be sunk for obtaining samples. For ore body/mineral at depth the samples are collected by sampling drill holes. For surface mining two maps are very important for this purpose which are mentioned below

1. **Topographical survey contour mapping-** Topographical survey is carried out and a contour with 1m to 2 m interval on a scale of 1 : 1000 to 1 : 2000 is prepared.
2. **Geological mapping-** In this map outcrops, structural features, ore body boundary and various types of mineral incorporates on a scale of 1 : 1000 to 1 : 2000 is prepared.

Trenches-

Trenches are excavated across the entyre deposit to remove the soil cover and expose the mineral body. It is also used to find out the lithological characteristics of the mineral body. Trenches are generally very long (may be more than a kilometer) sunk at a maximum depth of 2 m. It is generally 1.5m wide and kept at distance of 200 m apart. Sample are collected from the floor of the trenches at every 4m length for analyzing the quality of mineral. It also helps to know the contact rocks, presence of any disturbances, dirt capping, details of information of the ore body, etc. Trenches are generally dug at the right angle to the course of outcrops. Distance between the trenches is determined by the type of orebody and depth of soil. It is better to dug them at closer interval to determine the width and the value of outcrops.

Pits

Shallow pits-

On a grid pattern a large number of shallow pits of 2m to 3 m depth are sunk at suitable location to determine the extent of soil capping, quality and quantity of float ore (if any), the quality of ore exposed (by the analysis of the sample), the boundary of the deposit at the surface, and also to enable to study the contact rocks and disturbances. The cross section of the pit may be 1.2m to 2 m in length and 2 m in width.

Deep pits-

On a much wider grid pattern a small number deep pits of 8m to 12 m depth and the same cross section as mentioned in the 'shallow pits' are sunk to determine the typical characteristics of ore, thickness of overburden, dirt capping, quality of ore by bulk sampling by the pilot plant tests and also to obtain information of many other parameters for the interpretation of bore hole results.

Adits :

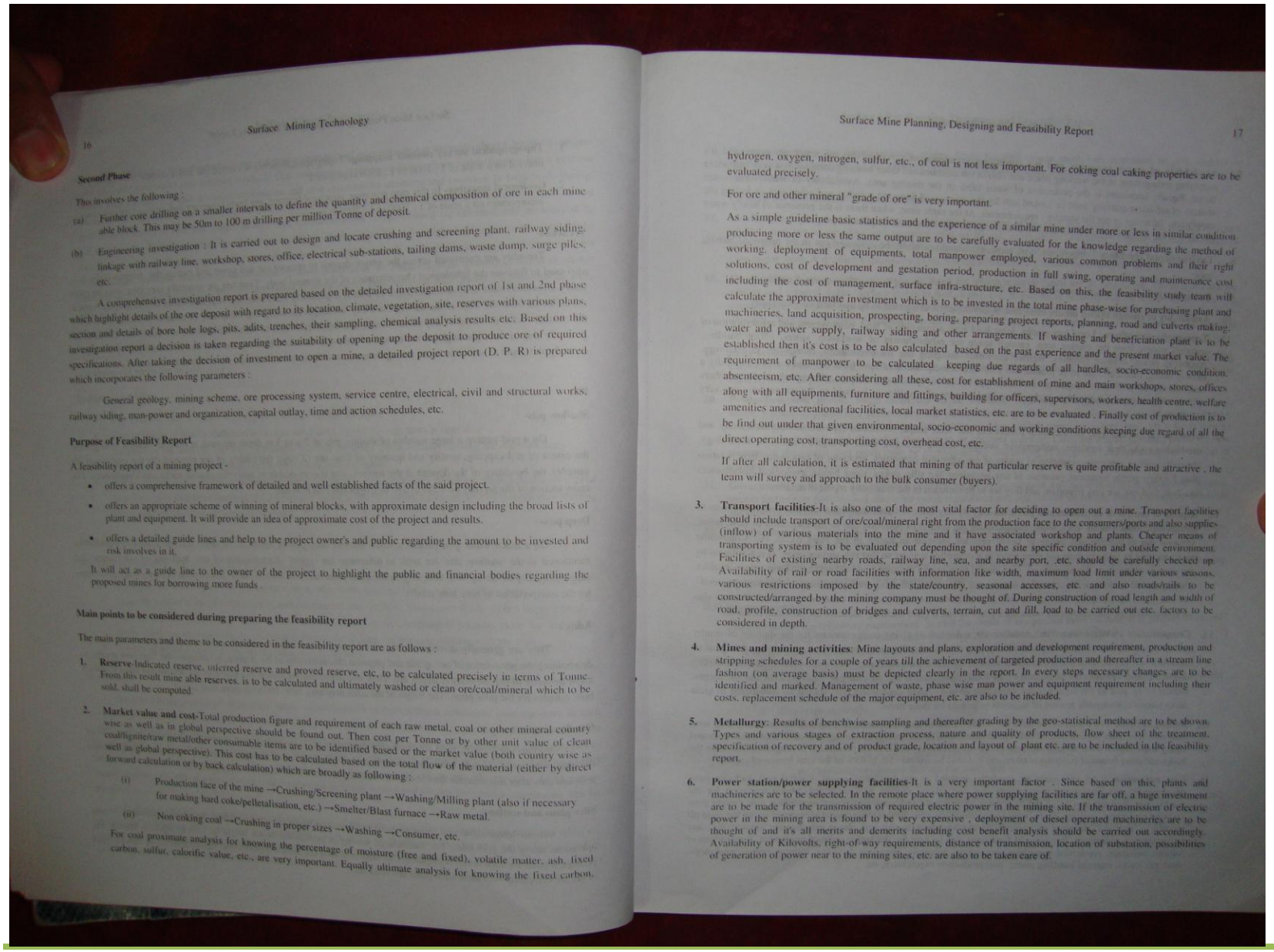
They are generally driven from the middle part of the deposit (sometimes driven in different parts also) for determining the persistence of ore, grade and physical characteristics of the deposit, to interpret the bore hole results, to know behavior of the ore body at depth, etc. They also determine the nature and extent of the deposit and assist to collect bulk samples for the pilot plant. The length of the adits may vary from 100 m to more than 200 m.

Ore dressing test-

Various ore dressing tests are conducted depending upon the end product required to determine the behavior of ore when subjected to various crushing, screening and washing operation. Composite samples are subjected to pilot plant tests and only after that exercise flow sheet for treatment of ROM (Run of mine) is designed.

Slice plans and sections-

Various longitudinal and cross sections are prepared based on results of bore hole and exploratory mining operations along the grid lines. For mechanized mining ,a provisional bench height of 10m to 40 m is chosen and geological plans for each bench is prepared. Calculation of total ore ,bench-wise ore reserve and reserves of various types of ore are computed from these slice plans and sections.



Second Phase

This involves the following :

(a) Further core drilling on a smaller intervals to define the quantity and chemical composition of ore in each mine able block. This may be 50m to 100 m drilling per million Tonne of deposit.

(b) Engineering investigation : It is carried out to design and locate crushing and screening plant, railway siding, linkage with railway line, workshop, stores, office, electrical sub-stations, tailing dams, waste dump, surge piles, etc.

A comprehensive investigation report is prepared based on the detailed investigation report of 1st and 2nd phase which highlight details of the ore deposit with regard to its location, climate, vegetation, site, reserves with various plans, section and details of bore hole logs, pits, adits, trenches, their sampling, chemical analysis results etc. Based on this investigation report a decision is taken regarding the suitability of opening up the deposit to produce ore of required specifications. After taking the decision of investment to open a mine, a detailed project report (D. P. R) is prepared which incorporates the following parameters :

General geology, mining scheme, ore processing system, service centre, electrical, civil and structural works, railway siding, man-power and organization, capital outlay, time and action schedules, etc.

Purpose of Feasibility Report

A feasibility report of a mining project -

- offers a comprehensive framework of detailed and well established facts of the said project.
- offers an appropriate scheme of winning of mineral blocks, with approximate design including the broad lists of plant and equipment. It will provide an idea of approximate cost of the project and results.
- offers a detailed guide lines and help to the project owner's and public regarding the amount to be invested and risk involves in it.

It will act as a guide line to the owner of the project to highlight the public and financial bodies regarding the proposed mines for borrowing more funds .

Main points to be considered during preparing the feasibility report

The main parameters and theme to be considered in the feasibility report are as follows :

1. **Reserve**-Indicated reserve, inferred reserve and proved reserve, etc, to be calculated precisely in terms of Tonne. From this result mine able reserves, is to be calculated and ultimately washed or clean ore/coal/mineral which to be sold, shall be computed.

2. **Market value and cost**-Total production figure and requirement of each raw metal, coal or other mineral country wise as well as in global perspective should be found out. Then cost per Tonne or by other unit value of clean coal/lignite/raw metal/other consumable items are to be identified based or the market value (both country wise as well as global perspective). This cost has to be calculated based on the total flow of the material (either by direct forward calculation or by back calculation) which are broadly as following :

   (i) Production face of the mine →Crushing/Screening plant →Washing/Milling plant (also if necessary for making hard coke/pelletalisation, etc.) →Smelter/Blast furnace →Raw metal.

   (ii) Non coking coal →Crushing in proper sizes →Washing →Consumer, etc.

   For coal proximate analysis for knowing the percentage of moisture (free and fixed), volatile matter, ash, fixed carbon, sulfur, calorific value, etc., are very important. Equally ultimate analysis for knowing the fixed carbon, hydrogen, oxygen, nitrogen, sulfur, etc., of coal is not less important. For coking coal caking properties are to be evaluated precisely.

   For ore and other mineral "grade of ore" is very important.

   As a simple guideline basic statistics and the experience of a similar mine under more or less in similar condition producing more or less the same output are to be carefully evaluated for the knowledge regarding the method of working, deployment of equipments, total manpower employed, various common problems and their right solutions, cost of development and gestation period, production in full swing, operating and maintenance cost including the cost of management, surface infra-structure, etc. Based on this, the feasibility study team will calculate the approximate investment which is to be invested in the total mine phase-wise for purchasing plant and machineries, land acquisition, prospecting, boring, preparing project reports, planning, road and culverts making, water and power supply, railway siding and other arrangements. If washing and beneficiation plant is to be established then it's cost is to be also calculated based on the past experience and the present market value. The requirement of manpower to be calculated keeping due regards of all hurdles, socio-economic condition, absenteeism, etc. After considering all these, cost for establishment of mine and main workshops, stores, offices along with all equipments, furniture and fittings, building for officers, supervisors, workers, health centre, welfare amenities and recreational facilities, local market statistics, etc. are to be evaluated . Finally cost of production is to be find out under that given environmental, socio-economic and working conditions keeping due regard of all the direct operating cost, transporting cost, overhead cost, etc.

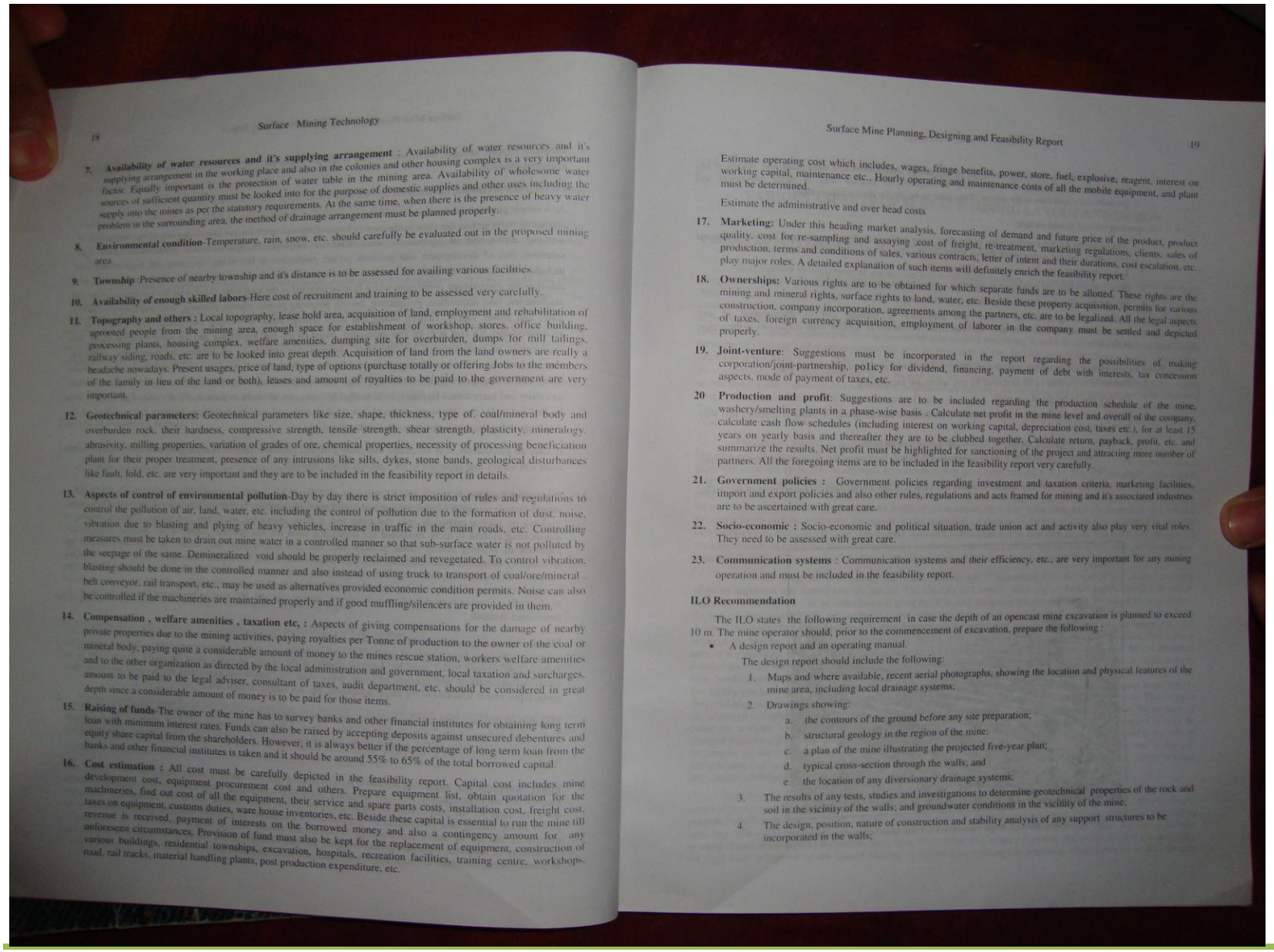
   If after all calculation, it is estimated that mining of that particular reserve is quite profitable and attractive , the team will survey and approach to the bulk consumer (buyers).

3. **Transport facilities**-It is also one of the most vital factor for deciding to open out a mine. Transport facilities should include transport of ore/coal/mineral right from the production face to the consumers/ports and also supplies (inflow) of various materials into the mine and it have associated workshop and plants. Cheaper means of transporting system is to be evaluated out depending upon the site specific condition and outside environment. Facilities of existing nearby roads, railway line, sea, and nearby port, etc. should be carefully checked up. Availability of rail or road facilities with information like width, maximum load limit under various seasons, various restrictions imposed by the state/country, seasonal accesses, etc. and also roads/rails to be constructed/arranged by the mining company must be thought of. During construction of road length and width of road, profile, construction of bridges and culverts, terrain, cut and fill, load to be carried out etc. factors to be considered in depth.

4. **Mines and mining activities**: Mine layouts and plans, exploration and development requirement, production and stripping schedules for a couple of years till the achievement of targeted production and thereafter in a stream line fashion (on average basis) must be depicted clearly in the report. In every steps necessary changes are to be identified and marked. Management of waste, phase wise man power and equipment requirement including their costs, replacement schedule of the major equipment, etc. are also to be included.

5. **Metallurgy**: Results of benchwise sampling and thereafter grading by the geo-statistical method are to be shown. Types and various stages of extraction process, nature and quality of products, flow sheet of the treatment, specification of recovery and of product grade, location and layout of plant etc. are to be included in the feasibility report.

6. **Power station/power supplying facilities**-It is a very important factor . Since based on this, plants and machineries are to be selected. In the remote place where power supplying facilities are far off, a huge investment are to be made for the transmission of required electric power in the mining site. If the transmission of electric power in the mining area is found to be very expensive , deployment of diesel operated machineries are to be thought of and it's all merits and demerits including cost benefit analysis should be carried out accordingly. Availability of Kilovolts, right-of-way requirements, distance of transmission, location of substation, possibilities of generation of power near to the mining sites, etc. are also to be taken care of.



7. **Availability of water resources and it's supplying arrangement** : Availability of water resources and it's supplying arrangement in the working place and also in the colonies and other housing complex is a very important factor. Equally important is the protection of water table in the mining area. Availability of wholesome water sources of sufficient quantity must be looked into for the purpose of domestic supplies and other uses including the supply into the mines as per the statutory requirements. At the same time, when there is the presence of heavy water problem in the surrounding area, the method of drainage arrangement must be planned properly.

8. **Environmental condition**-Temperature, rain, snow, etc. should carefully be evaluated out in the proposed mining area.

9. **Township** :Presence of nearby township and it's distance is to be assessed for availing various facilities.

10. **Availability of enough skilled labors**-Here cost of recruitment and training to be assessed very carefully.

11. **Topography and others :** Local topography, lease hold area, acquisition of land, employment and rehabilitation of uprooted people from the mining area, enough space for establishment of workshop, stores, office building, processing plants, housing complex, welfare amenities, dumping site for overburden, dumps for mill tailings, railway siding, roads, etc. are to be looked into great depth. Acquisition of land from the land owners are really a headache nowadays. Present usages, price of land, type of options (purchase totally or offering Jobs to the members of the family in lieu of the land or both), leases and amount of royalties to be paid to the government are very important.

12. **Geotechnical parameters:** Geotechnical parameters like size, shape, thickness, type of coal/mineral body and overburden rock, their hardness, compressive strength, tensile strength, shear strength, plasticity, mineralogy, abrasivity, milling properties, variation of grades of ore, chemical properties, necessity of processing beneficiation plant for their proper treatment, presence of any intrusions like sills, dykes, stone bands, geological disturbances like fault, fold, etc. are very important and they are to be included in the feasibility report in details.

13. **Aspects of control of environmental pollution**-Day by day there is strict imposition of rules and regulations to control the pollution of air, land, water, etc. including the control of pollution due to the formation of dust, noise, vibration due to blasting and plying of heavy vehicles, increase in traffic in the main roads, etc. Controlling measures must be taken to drain out mine water in a controlled manner so that sub-surface water is not polluted by the seepage of the same. Demineralized void should be properly reclaimed and revegetated. To control vibration, blasting should be done in the controlled manner and also instead of using truck to transport of coal/ore/mineral , belt conveyor, rail transport, etc., may be used as alternatives provided economic condition permits. Noise can also be controlled if the machineries are maintained properly and if good muffling/silencers are provided in them.

14. **Compensation , welfare amenities , taxation etc, :** Aspects of giving compensations for the damage of nearby private properties due to the mining activities, paying royalties per Tonne of production to the owner of the coal or mineral body, paying quite a considerable amount of money to the mines rescue station, workers welfare amenities and to the other organization as directed by the local administration and government, local taxation and surcharges, amount to be paid to the legal adviser, consultant of taxes, audit department, etc. should be considered in great depth since a considerable amount of money is to be paid for those items.

15. **Raising of funds**-The owner of the mine has to survey banks and other financial institutes for obtaining long term loan with minimum interest rates. Funds can also be raised by accepting deposits against unsecured debentures and equity share capital from the shareholders. However, it is always better if the percentage of long term loan from the banks and other financial institutes is taken and it should be around 55% to 65% of the total borrowed capital.

16. **Cost estimation :** All cost must be carefully depicted in the feasibility report. Capital cost includes mine development cost, equipment procurement cost and others. Prepare equipment list, obtain quotation for the machineries, find out cost of all the equipment, their service and spare parts costs, installation cost, freight cost, taxes on equipment, customs duties, ware house inventories, etc. Beside these capital is essential to run the mine till revenue is received, payment of interests on the borrowed money and also a contingency amount for any unforeseen circumstances. Provision of fund must also be kept for the replacement of equipment, construction of various buildings, residential townships, excavation, hospitals, recreation facilities, training centre, workshops-road, rail tracks, material handling plants, post production expenditure, etc.

Estimate operating cost which includes, wages, fringe benefits, power, store, fuel, explosive, reagent, interest on working capital, maintenance etc., Hourly operating and maintenance costs of all the mobile equipment, and plant must be determined.

Estimate the administrative and over head costs

17. **Marketing:** Under this heading market analysis, forecasting of demand and future price of the product, product quality, cost for re-sampling and assaying ,cost of freight, re-treatment, marketing regulations, clients, sales of production, terms and conditions of sales, various contracts, letter of intent and their durations, cost escalation, etc. play major roles. A detailed explanation of such items will definitely enrich the feasibility report.

18. **Ownerships:** Various rights are to be obtained for which separate funds are to be allotted. These rights are the mining and mineral rights, surface rights to land, water, etc. Beside these property acquisition, permits for various construction, company incorporation, agreements among the partners, etc. are to be legalized. All the legal aspects of taxes, foreign currency acquisition, employment of laborer in the company must be settled and depicted properly.

19. **Joint-venture**: Suggestions must be incorporated in the report regarding the possibilities of making corporation/joint-partnership, policy for dividend, financing, payment of debt with interests, tax concession aspects, mode of payment of taxes, etc.

20 **Production and profit**: Suggestions are to be included regarding the production schedule of the mine, washery/smelting plants in a phase-wise basis . Calculate net profit in the mine level and overall of the company, calculate cash flow schedules (including interest on working capital, depreciation cost, taxes etc.), for at least 15 years on yearly basis and thereafter they are to be clubbed together. Calculate return, payback, profit, etc. and summarize the results. Net profit must be highlighted for sanctioning of the project and attracting more number of partners. All the foregoing items are to be included in the feasibility report very carefully.

21. **Government policies :** Government policies regarding investment and taxation criteria, marketing facilities, import and export policies and also other rules, regulations and acts framed for mining and it's associated industries are to be ascertained with great care.

22. **Socio-economic :** Socio-economic and political situation, trade union act and activity also play very vital roles. They need to be assessed with great care.

23. **Communication systems** : Communication systems and their efficiency, etc., are very important for any mining operation and must be included in the feasibility report.

ILO Recommendation

The ILO states the following requirement in case the depth of an opencast mine excavation is planned to exceed 10 m. The mine operator should, prior to the commencement of excavation, prepare the following :

- A design report and an operating manual.

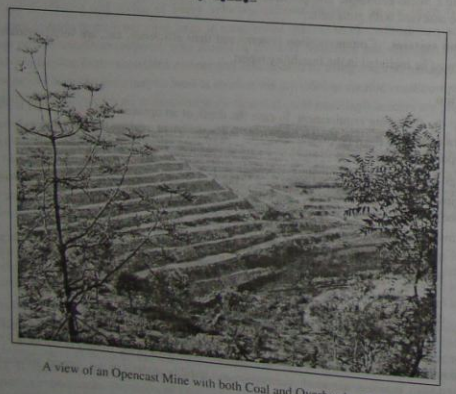
The design report should include the following:

1. Maps and where available, recent aerial photographs, showing the location and physical features of the mine area, including local drainage systems;
2. Drawings showing:
   a. the contours of the ground before any site preparation;
   b. structural geology in the region of the mine;
   c. a plan of the mine illustrating the projected five-year plan;
   d. typical cross-section through the walls; and
   e. the location of any diversionary drainage systems;
3. The results of any tests, studies and investigations to determine geotechnical properties of the rock and soil in the vicinity of the walls; and groundwater conditions in the vicinity of the mine;
4. The design, position, nature of construction and stability analysis of any support structures to be incorporated in the walls;

5. A stability analysis including an outline of any design assumptions that were made and details of any testing and instrumentation which may be required to monitor wall or strata movement and groundwater conditions in the region of the mine to verify the design assumptions.

- The operating manual should include:
  1. Specifications for the configuration of the final mine walls, including toe and crest positions, planned face slopes, average overall wall slopes, bench widths, bench intervals and any berm construction;
  2. The method and rate of excavation to the final walls;
  3. Details of any blasting procedure to maintain control of the pit wall;
  4. Specifications for drainage and dewatering systems; and
  5. Details including location, method and frequency of reading and of maintenance of any instrumentation required to monitor wall and strata movement or groundwater conditions in the region of the mine.

- Where an amendment to any design is proposed, the operator should prepare a report showing:
  1. The extent of the land to be disturbed by any additional excavation;
  2. The new design height of any wall;
  3. Details of any additional site preparation and excavation, including any blasting procedures that may be required; and
  4. An analysis of the safety aspects of the intended design to include:
     a. the results of geotechnical studies, including drill hole logs, tests and groundwater measurements; and
     b. the results of stability studies to assess any effects of the amendment on the security of the walls.



A view of an Opencast Mine with both Coal and Overburden Benches

CHAPTER 2

VARIOUS ASPECTS OF SURFACE MINING LAYOUT AND DESIGN

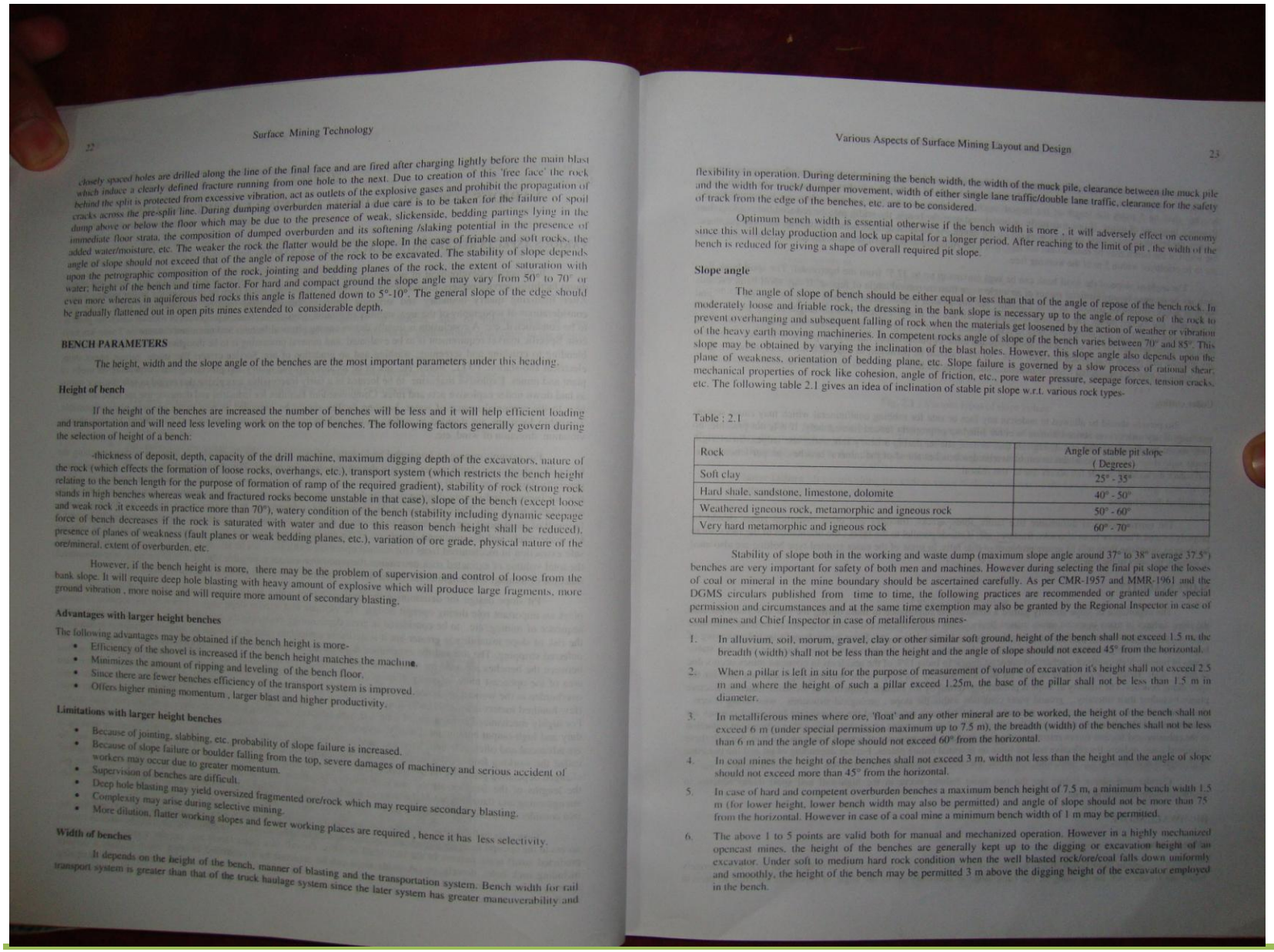
VARIOUS ASPECTS REGARDING SURFACE MINING LAYOUT

It comprises development of production and overburden benches, overburden dump facility, waste and reject dump facilities, service centers, township, water resources, connecting roads from pit to processing and/or loading yards. After finalizing quarry boundary and stripping limits processing plant location is determined after taking due considerations of topography of the area, stability of ground, physico-mechanical properties of rock on which the plant is to be constructed, etc. A conclusion is usually drawn among physical features and minimum distance on Tonne-km haul cost. Specific market requirement is to be evaluated ,and mineral processing is to be thought of accordingly from simple blending to crushing and screening, washing and up gradation of ore. Service centers like stores, garage, repair shop, electrical sub-station, P.O.L. stores, etc. are to be located between quarry and processing plant depending on needs of plant and mines. Explosive magazine to be located in a fairly inaccessible area giving due regard to safe distance norms as laid down under explosive acts and rules. Compressed air facilities for industrial and domestic use are to be developed near to the perennial source of water. Loading facility should be provided as near to the crushing plant as possible. Township locations are decided according to the architectural considerations as well as considering blasting shocks, vibration, direction of wind, etc.

For openpit mining in horizontal and gently dipping deposit, the overburden is removed without touching the footwall and ultimate depth varies slightly during the operation period. The ultimate depth of modern opencast mines generally designed up to 200 m although sometimes it may goes even beyond 400 m. The ultimate depth plays a great role for inclined and steeply dipping deposits for their production capacity, surface geometry, the actual volume of the overburden rocks, etc. The dimension of the openpit mine is determined by the size of the deposit, the depth, pit slope, footwall of the deposit, etc. The length and width of the opencast mine may sometimes exceeds more than 10 km and 5 km respectively and the minimum dimension of the openpit floor should not be less than 50 m long and 35 m wide for safe extraction in mechanized faces (for manual operation width shall not be less than 10 m). The reserve of the ore and the total volume of extracted rock determines the production capacity, possible output, life of the mine, and economical outcome of mining. The length of the bench should be at least 3 times the blasting length done at a time.

Pit slope design for determining the final pit layout which maximizes financial returns and mineral recovery plays an important role during openpit design. The ultimate and interim slope angles, bench angles, location of ramps, sequence of mining, etc. to be considered in great depth during designing the openpit layout. Although in steep slopes, the risk of slope instability is greater but it is desirable sometimes since the volume of waste excavation is reduced and deferred stripping. The ore and the overburden is extracted in descending slicing (bench shape) and the bench floor left between the benches are used for accommodating shovel, drills, roads for dumpers and for other purposes. The surface area of the opencast mine in gently dipping deposits are increased and back filling is generally done by the extracted overburden in the worked out space whereas in case of thick steeply dipping deposits they are extracted up to great depth (few hundred meters deep) and backfilling in the demineralized area by the extracted overburden is generally not done. For highly mechanized openpit mines long and high benches, wide bench floor, etc. are needed to equip large, heavy-duty and high-output equipment. Usually, several benches are worked simultaneously in an openpit mine. The benches are advanced and often new benches are created over the openpit depth. The length of a bench developed for working is called the working front of the bench, which is straight line or curvilinear in plan .It may remain constant or may be changed depending primarily on the shape and size of the deposit. The length of the opencast mining front is the sum of the lengths of the front of all the working benches. Measures for draining out water from surface and in the pit, strengthening and flattening out of the slope of benches, caving or slope failure, fire and dust control, etc., are to be taken into consideration during the openpit layout.

Generally individual benches are steeper than the overall face. Bench floor (shown in the Fig. 1.1) provides the access to the openpit face, act as rock traps and during designing bench width due consideration are to be taken to control predicted small scale failures in the individual benches. Overall face (final faces) may require stabilization measures including rock bolts, dowels, etc. A stable slope can be achieved by a controlled blasting which fracture a definite quantity of rock leaving intact and undisturbed faces. Over break and induced fractures will be minimized. For designing the final faces in openpit mines special blasting technique (pre-splitting) is practiced. Here a row of small-diameter



closely spaced holes are drilled along the line of the final face and are fired after charging lightly before the main blast which induce a clearly defined fracture running from one hole to the next. Due to creation of this 'free face' the rock behind the split is protected from excessive vibration, act as outlets of the explosive gases and prohibit the propagation of cracks across the pre-split line. During dumping overburden material a due care is to be taken for the failure of spoil dump above or below the floor which may be due to the presence of weak, slickenside, bedding partings lying in the immediate floor strata, the composition of dumped overburden and its softening /slaking potential in the presence of added water/moisture, etc. The weaker the rock the flatter would be the slope. In the case of friable and soft rocks, the angle of slope should not exceed that of the angle of repose of the rock to be excavated. The stability of slope depends upon the petrographic composition of the rock, jointing and bedding planes of the rock, the extent of saturation with water; height of the bench and time factor. For hard and compact ground the slope angle may vary from 50° to 70° or even more whereas in aquiferous bed rocks this angle is flattened down to 5°-10°. The general slope of the edge should be gradually flattened out in open pits mines extended to considerable depth.

BENCH PARAMETERS

The height, width and the slope angle of the benches are the most important parameters under this heading.

Height of bench

If the height of the benches are increased the number of benches will be less and it will help efficient loading and transportation and will need less leveling work on the top of benches. The following factors generally govern during the selection of height of a bench:

-thickness of deposit, depth, capacity of the drill machine, maximum digging depth of the excavators, nature of the rock (which effects the formation of loose rocks, overhangs, etc.), transport system (which restricts the bench height relating to the bench length for the purpose of formation of ramp of the required gradient), stability of rock (strong rock stands in high benches whereas weak and fractured rocks become unstable in that case), slope of the bench (except loose and weak rock ,it exceeds in practice more than 70°), watery condition of the bench (stability including dynamic seepage force of bench decreases if the rock is saturated with water and due to this reason bench height shall be reduced), presence of planes of weakness (fault planes or weak bedding planes, etc.), variation of ore grade, physical nature of the ore/mineral, extent of overburden, etc.

However, if the bench height is more, there may be the problem of supervision and control of loose from the bank slope. It will require deep hole blasting with heavy amount of explosive which will produce large fragments, more ground vibration , more noise and will require more amount of secondary blasting.

Advantages with larger height benches

The following advantages may be obtained if the bench height is more-

- Efficiency of the shovel is increased if the bench height matches the machine.
- Minimizes the amount of ripping and leveling of the bench floor.
- Since there are fewer benches efficiency of the transport system is improved.
- Offers higher mining momentum , larger blast and higher productivity.

Limitations with larger height benches

- Because of jointing, slabbing, etc. probability of slope failure is increased.
- Because of slope failure or boulder falling from the top, severe damages of machinery and serious accident of workers may occur due to greater momentum.
- Supervision of benches are difficult.
- Deep hole blasting may yield oversized fragmented ore/rock which may require secondary blasting.
- Complexity may arise during selective mining.
- More dilution, flatter working slopes and fewer working places are required , hence it has less selectivity.

Width of benches

It depends on the height of the bench, manner of blasting and the transportation system. Bench width for rail transport system is greater than that of the truck haulage system since the later system has greater maneuverability and

flexibility in operation. During determining the bench width, the width of the muck pile, clearance between the muck pile and the width for truck/ dumper movement, width of either single lane traffic/double lane traffic, clearance for the safety of track from the edge of the benches, etc. are to be considered.

Optimum bench width is essential otherwise if the bench width is more , it will adversely effect on economy since this will delay production and lock up capital for a longer period. After reaching to the limit of pit , the width of the bench is reduced for giving a shape of overall required pit slope.

Slope angle

The angle of slope of bench should be either equal or less than that of the angle of repose of the bench rock. In moderately loose and friable rock, the dressing in the bank slope is necessary up to the angle of repose of the rock to prevent overhanging and subsequent falling of rock when the materials get loosened by the action of weather or vibration of the heavy earth moving machineries. In competent rocks angle of slope of the bench varies between 70° and 85°. This slope may be obtained by varying the inclination of the blast holes. However, this slope angle also depends upon the plane of weakness, orientation of bedding plane, etc. Slope failure is governed by a slow process of rational shear; mechanical properties of rock like cohesion, angle of friction, etc., pore water pressure, seepage forces, tension cracks, etc. The following table 2.1 gives an idea of inclination of stable pit slope w.r.t. various rock types-

Table : 2.1

Rock	Angle of stable pit slope (Degrees)
Soft clay	25° - 35°
Hard shale, sandstone, limestone, dolomite	40° - 50°
Weathered igneous rock, metamorphic and igneous rock	50° - 60°
Very hard metamorphic and igneous rock	60° - 70°

Stability of slope both in the working and waste dump (maximum slope angle around 37° to 38° average 37.5°) benches are very important for safety of both men and machines. However during selecting the final pit slope the losses of coal or mineral in the mine boundary should be ascertained carefully. As per CMR-1957 and MMR-1961 and the DGMS circulars published from time to time, the following practices are recommended or granted under special permission and circumstances and at the same time exemption may also be granted by the Regional Inspector in case of coal mines and Chief Inspector in case of metalliferous mines-

1. In alluvium, soil, morum, gravel, clay or other similar soft ground, height of the bench shall not exceed 1.5 m, the breadth (width) shall not be less than the height and the angle of slope should not exceed 45° from the horizontal.
2. When a pillar is left in situ for the purpose of measurement of volume of excavation it's height shall not exceed 2.5 m and where the height of such a pillar exceed 1.25m, the base of the pillar shall not be less than 1.5 m in diameter.
3. In metalliferous mines where ore, 'float' and any other mineral are to be worked, the height of the bench shall not exceed 6 m (under special permission maximum up to 7.5 m), the breadth (width) of the benches shall not be less than 6 m and the angle of slope should not exceed 60° from the horizontal.
4. In coal mines the height of the benches shall not exceed 3 m, width not less than the height and the angle of slope should not exceed more than 45° from the horizontal.
5. In case of hard and competent overburden benches a maximum bench height of 7.5 m, a minimum bench width 1.5 m (for lower height, lower bench width may also be permitted) and angle of slope should not be more than 75 from the horizontal. However in case of a coal mine a minimum bench width of 1 m may be permitted.
6. The above 1 to 5 points are valid both for manual and mechanized operation. However in a highly mechanized opencast mines, the height of the benches are generally kept up to the digging or excavation height of an excavator. Under soft to medium hard rock condition when the well blasted rock/ore/coal falls down uniformly and smoothly, the height of the bench may be permitted 3 m above the digging height of the excavator employed in the bench.

The width of the benches shall not be less than the widest machine plying on the bench plus 2 m more or 3 times the width of the dumper if dumpers ply on the bench or the height of the bench whichever is more, or the width of the benches shall be 5 times the length of the largest machine working on the bench unless otherwise permitted by the Regional Inspector of mines. If one way traffic is introduced over the bench, the width may be reduced but not less than 2 times the length of the largest machine working on the bench. At the final operation during making the final pit slope, the width of the bench may be reduced but not less than 3 m. Adequate dressing of the benches are to be done if persons are to be employed within 5 m of the working face.

The angle of slope of the spoil bank can be kept maximum up to 37.5° from the horizontal. The spoil bank face shall not be retained by any artificial means at an angle more than its natural angle of repose. If any spoil bank exceeds 30 m in height, it shall be benched in such a manner that the general overall slope of spoil does not exceed 33.7° and height does not exceed 30 m. The toe of a spoil bank shall not be nearer by less than its height to any public road, railway, public buildings and other permanent structure which are not belonging to the mines owner and a suitable fence shall be erected between the toe of an active spoil bank and those of public roads, railways or public structures to prevent any unauthorized person to approach the spoil bank. No body should be permitted to approach the toe of an active spoil bank where he may be endangered from the material rolling down the face.

Under cutting

No persons should be allowed to undercut any face or side for robbing coal/mineral which may cause serious overhang. If any under cut is noticed it must be either filled up or properly fenced immediately. If it is not possible, no body should be allowed to take rest during rest time or take shelter during a heavy rain under the ledges. Strict vigil, proper supervision, keeping the advancement of overburden benches ahead of the mineral benches, proper fencing, etc. will reduce the accident of side fall occurs due to this reason.

Geological disturbances

The entyre geological disturbance like faults, folds, cracks, fractures and other disturbed zones should be properly located. Planes or faces of cracks, fractures, etc. if stands parallel to the faces , the orientation of the faces are to be changed, otherwise there will be severe problems of side falls. In some of the cases grouted type bolting are also used to stabilize the sides of the benches.

SLOPE STABILITY

The importance of slope stability gained a considerable importance in India during the last few years because the slope failures in many opencast mines caused heavy losses of production, extra stripping cost to remove land-slide material, reforming of benches, diversion of haul roads ,production delays ,etc. Since heavy benefit including saving of huge amount of money is obtained from the steepest slope, the openpit designer now a days focusing on steep stable slopes in running high- walls of mechanized opencast mines. In India 18% of the accidents in opencast mines occurs due to fall of sides .Hence due care is to be paid for the slope stability not only for the working high-wall but also for the spoil dump benches .The stability of slope is primarily governed by the strength properties of the slope materials,fracture planes including their intensity , ground water condition within the slope , geological structures , weathering effects , ground vibration etc. As the excavation proceeds the vertical stress reduces and the pre-existing horizontal stresses are induced beneath the pit bottom and around the pit ends. This causes opening of cracks/ joints with a subsequent reduction in the cohesive and friction forces restraining the rock in place. The presence of ground water and its flow through these distressed zones reduces the effective normal force on potential failure planes. Further depending of pit and the presence of geological disturbances e.g. fault, fold, dykes, etc. the distressed zone will be enhanced.

Circular and planar failure are the two basic failures occur in the high-wall and spoil dump benches in weathered slope material in India.

HIGHWALL SLOPE

Slope Failures

The angle of slope bench should be either equal or less than the angle of repose of the bench rock. In moderately loose and friable rock the dressing in the bank slope is necessary up to the angle of repose of the rock to prevent

overhanging and subsequent falling of rock when the materials get loosened by the action of weather or vibration of the heavy earth moving machinery. In competent rocks angle of slope of the bench varies between 70° and 85°. This slope may be obtained by varying the inclination of the blast holes. However, this slope angle also depends upon the plane of weakness, orientation of bedding planes ,etc. Slope failure is mainly due to the slow process of rotational shear. mechanical properties of rock (like cohesion, angle of internal friction, etc., pore water pressure, seepage forces, tension cracks etc). The various types of slope failures are shown in the Fig. 2.1

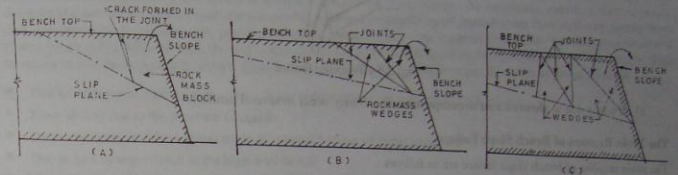


Fig. 2.1 : Various types of slope failure.

The height of the dragline benches may even go beyond 50m but in Indian condition it is restricted within 40m. Common dragline bench height starts from 30m (for draglines having bucket capacity 24m^3 with a boom length 96m). The bench slope failure may be broadly grouped into five categories which are as follows :

a) **Planar failure** : It occurs along the various types of discontinuities on the bench scale, inter ramp scale and pit wall scale (e. g. major fault) Fig. 2.2.

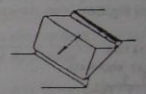


Fig. 2.2 : Planar failure

b) **Circular failure** : This type of deep seated failures occur when a slope is excavated in soil or soft rock formation in which the mechanical properties are not dominated by clearly defined structural features (Fig.2.3).

Fig. 2.3 : Circular failure

c) **Wedge failure** : It is due to two intersecting discontinuities. Large scale wedge failure may occur in several benches (Fig.2.4).

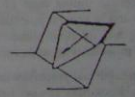


Fig. 2.4 : Wedge failure

d) **Toppling failure** : It is generally occurs in hard rock formations which can form columnar structure separated by steeply dipping discontinuities (Fig.2.5).

Fig. 2.5 : Toppling failure

e) Failure due to **sheared and decomposed** rock: It may occur in several benches.

The Main Reasons of Bench Slope Failure

The main reasons of bench slope failure are as follows :

- Dynamic loading caused by plying HEMM, blasting, earthquakes etc. Shear stresses are momentarily increased as a result of vibration in the rock mass.
- Water pressure in the joint has probably been responsible for more frequent slope failure than all other causes taken together. Hence a thorough knowledge of the character and influence of the hydrologic regime is necessary. Physical and chemical effects of pore water pressure in joint in filling materials can thus alter the cohesion- friction parameters. Physical effects of providing uplift on potential sliding and exerting joint water (pore water) pressure on the joint surfaces reduce the frictional resistance. This will reduce the shearing resistance along the potential failure plane by reducing the effective normal stress acting on it. Physical and chemical effects of water pressure in the intact rock cause a decrease in compressive strength particularly where confining stress has been reduced. Precipitation also causes bench slope failure (Fig. 2.6).

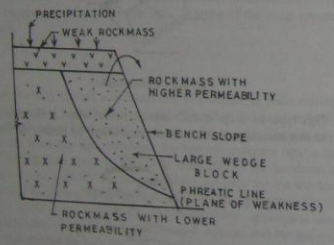


Fig. 2.6 : Effect of precipitation on rock mass and growing of large wedge block for failure

- The most serious problem in rock slope failure often is concerned with the location, orientation and properties of structural discontinuities in the rock-mass. Structural discontinuities control the mechanical response and hence the stability of rock slope. The geometry of potential sliding mass of an unstable rock slope is determined by the location and orientation of the discontinuities and in general potential failure surfaces are dictated by the structural weakness such as fault zone, joints, bedding and foliation planes. Due to the higher height amount of joining, occurrence of slabbing, etc. shall be more and probability of slope failure shall definitely be high.

Rockmass is an isotropic in nature since it's strength and properties are variable and dependent on attitude, geometry and spatial distribution of the structural defects. Rockmass is also heterogeneous in nature since it's physical and lithological properties are variable. To detect the structural discontinuity the joint , factors affecting shear strength on the joint, factors affecting shear along a potential plane of weakness in the slopes, etc. are very important.

Geological discontinuities have overriding importance in the stability of the bench slope. A structural discontinuity associated with both the incipient plane of failure and the plane along which the movement of the rock mass ultimately takes place. Next to orientation, position, intensity of joint sets it is very important. Joint sets are randomly distributed in the rock mass and their surface as purities (roughness, waviness etc.) are responsible for the movement characteristics and strength properties of the joint. The various types of geological structure (faults, fold, tension cracks, bedding planes, etc.) have different geometry and spatial distribution. Because of these reasons the roughness, nature of their contacts, degree and nature of weathering type and amount of gauge and susceptibility of ground water flow will vary greatly from one structural type to another. The presence of gauge materials (it is the material that occurs between two structural planes which is the different from the hard rock) causes the increase and decrease in resistance of sliding surfaces. This increases or decreases in sliding surface depends upon the nature, type, character and thickness of gauge material.

- Due to submergence of slope in water : Water reduces the shear strength of rock mass.
- Floor sliding due to the presence of aquifer.
- Due to decrease in cohesion and angle of internal friction of dump materials.
- Due to lack of supervision in the high-wall bench.
- If the deep hole blasting is not done properly, there shall be the chances of formation of ledges in the high-wall and the chances of slope failure becomes high.
- In deep-hole blasting maintenance of slope angle is also very difficult and probability of slope failure becomes very high.

FACTOR OF SAFETY OF BENCH SLOPE

The safety factor is defined as the ratio of the maximum force which resists sliding and the existing force which tends to induce sliding. In other word the safety factor is defined as the ratio between the maximum shear strength on the examined potential sliding surface and the effectively mobilized shear stress.

$$\text{Factor of Safety } (F_s) = \frac{\textit{Maximum mobilitable shear strength}}{\textit{Effectivr mobilized shear stress}}$$

The only force acting on the sliding block is the gravity, hence the block weight force M acts vertically downwards (Fig 2.7).

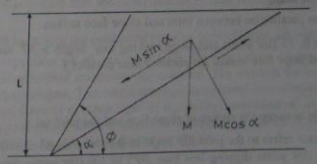


Fig. 2.7

Let M is the weight of the sliding block.

The component of M which tends to cause the sliding of the block is $M \sin \alpha$. The component of M perpendicular to the contact surface between the base of the block and the inclined plane is

$M \cos\alpha$. The normal stress σ acting across the contact is $\sigma = (M \cos \alpha)/A$,

where A is the area of the block base and α is the angle of discontinuity from the horizontal.

Assuming that the shear strength on the sliding surface is defined by the Coulomb criterion

$\tau = c + \sigma \tan \varphi$, where c and φ are the cohesion and angle of friction respectively.

Hence shear strength of the block:

$\tau = c + M \cos \alpha \tan \varphi / A$.

The shear force F ($F = \tau A$) which resists to sliding down the inclined plane is given by

$F = cA + M \cos\alpha \tan\varphi$

The limiting block equilibrium condition corresponds to the perfect equilibrium between the forces which tend to cause the block to slide and the forces which tend to oppose the sliding of the block.

$M \sin\alpha = cA + M \cos\alpha \tan\varphi$

If cohesion = 0, then $\alpha = \varphi$

The safety factor is defined as the ratio of the maximum mobilized force which resists sliding and the existing force which tends to induce sliding. In terms of stresses, the safety factor is defined as the ratio between the maximum mobilizable shear strength on the examined potential sliding surface and the effectively mobilized shear stress.

A high F.O.S causes low risk of failure by flattening the slope angle. But it will increase the volume of excavation of overburden rock and hence cost. Therefore, the F.O.S must be the optimum.

A F.O.S. shall be more than 1 and within 1.5 to 1.7 to minimize the risk of slope failure. The F.O.S. is taken more than 1 due to the reasons as mentioned below:-

- The possible presence of geological disturbances within the bench slope
- The variability of the water pressure and properties of rock within the bench.
- The errors on testing the geotechnical properties of rock and also errors in calculation of optimum slope angle.

The angle of bench slope shall always be more than the critical slope angle. At the critical slope angle, the F.O.S. of slope of the bench becomes 1. The slope angle shall be flattered 5^0 to 6^0 more compare to a critical slope angle so that F.O.S. falls in between 1.5 to 1.7.

Precautionary Measures to be Taken Against Slope Failure

Before designing any bench height and it's slope angle, the slope mass rating (SMR) should be obtained from RMR by subtracting a factorial adjustment factor depending on the joint-slope relationship and adding a factor depending on the method of excavation .

$SMR = RMR + (S_1 \times S_2 \times S_3) + S_4$

Where RMR = Rock mass rating

S_1 = A factor depends on parallelism between joint and slope face strikes.

It's value ranging from 0. 15 (for joints and slope making any angle > 30^0 and the failure probability is very low.) to 1.0 (when both joint and slope face strikes are parallel/ near parallel).

Relationship : $S_1 = (1 - \text{Sin } A)^2$

Where A = Angle between the strikes of slope face and the joint set.

S_2 = A factor refers to the joint dip angle in the shear mode of failure. It's value also ranging from 0.15 (for joints dipping less than 20°) to 1.0 (for joints dipping more than 45°) in case of planer failure.

S_2 = $\text{Tan}^2(\gamma_j)$, where (γ_j) represents joint dip angle. In case of toppling failure, value of S_2 becomes 1 in all dip angles.

S_3 = A factor depends upon the relationship between the slope face and joint dip. In case of planer failure the bench slope dip amount to be subtracted from the joint dip amount. Whereas in case of toppling failure bench slope dip amount and joint dip amount to be added up

S_4 = A factor of adjustment depends upon the protection management controlled blasting technique (e.g. pre-splitting, smooth blasting etc.)

Value of S_4 for built in protection management +15, for pre-splitting +10, for smooth wall blasting +8, mechanical excavation 0 , inefficient blasting -8, etc.

The Table -2.2 depicts the SMR classes and the slope stability.

- The geo-mechanical information which are most relevant to the slope stability are shear strength, or cohesive strength, the angle of friction and the density of rockmass. Determination of shear strength of intact rock and along discontinuities in rockmass is critical part of any slope design.
- The angle of slope of bench should be either equal or less than the angle of repose of the bench rocks . In competent rocks angle of slope of the high-wall benches varies between 65°-80° .The slope angle depends upon the plane of weakness, orientation of bedding planes, etc. However the slope wall shall not be retained by the artificial means at an angle in excess of it's natural angle of repose.
- Discontinuities in the rockmass should, therefore, be observed firstly w.r.t. their orientation, geometry and special distribution (Fig. 2.8). Particular attention must be given to the relationship with the direction and inclination of the rock slope and to any factors which might influence any potential plane or combination of plane or surface failure However, those dipping towards the slope must be carefully considered.

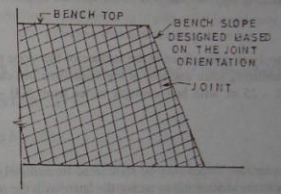


Fig. 2.8 : Bench slope design according to the joint in the rock-mass.

- Since the height of the dragline benches are very high, probability of presence of plane of weakness (fracture plane, geological disturbances, etc.) , seepage of water, increasing of ground vibration due to blasting, decreasing of cohesion and angle of internal friction of the bench rock mass are very acute and hence bench slope angle to be decreased a little accordingly provided that the other factors remain unchanged and the same factor of safety are maintained.
- Bench slope may be stabilized by practicing controlled blasting (pre-splitting, smooth wall blasting etc.)
- A high degree of regular inspection of all lhe high-walls by appropriate means (telescope, modern stability instruments, etc.) and appropriate trained manpower are essential. The unstable surface or edges of high-walls are to be designated by flags/ pegs/posts/signs. The mines manager/nominated executive/competent person shall monitor all the critical locations, defect, fissures or other signs of instability, conditions of all the catchments areas, etc. and shall arrange to make and keep the bench slope in secured condition. The density of inspection shall be heavy after a long periods of continuous rain or after torrential downpour of rain.

Supporting of Slopes

Supporting of the high-wall slopes may control the failure of slopes. The followings are the main supporting systems for high-wall slopes:

Toe ditches : They are very useful to keep the fallen rocks out of the road when failure are in the form of wedges, planes and minor topples.

Fences : They may be placed at the toe or in the slope to prevent the slope failure.

Nets : Nets may be placed over the slope plain to avoid free fall of rock piles. The nets are very much useful to prevent wedge failure and minor topples.

Bolting : Bolts in slopes are used as an immediate and also as a permanent supports. Both the tensioned (split and wedge type bolts, length 3m to 4 m, dia. 22 mm, strength 120 – 150 kN), and untensioned (normally resin/cement grouted type bolts, length 1 - 4 m, dia. 22 – 25 mm) are used with various bolting densities and layouts depending upon the slope mass ratings (Fig. 2.9)

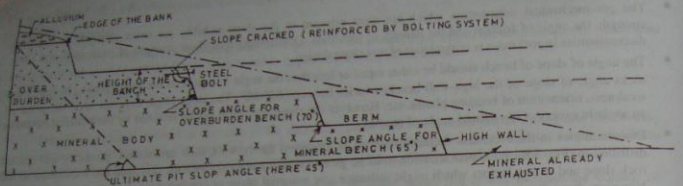


Fig. 2.9 : Face bolting for reinforcement in the cracked slope

Anchoring :

They are very long steel bars which apply an active force in the surface of the slope transferring it to the ground behind the unstable zone. They introduce a stabilizing force and also at the same time increase the shear strength in the joints. Anchor length generally varies from 8 – 25 m, with bolt strength varies from 500 – 2000 kN, and the anchor density varies from 10 – 30 number per m^2.

Shotcreting :

When the first signs of instability appear, the slopes may be shotcreted immediately. If necessary wire netting /slope bolting may combindly be used along with the shotcreting to secure the latter with the rockmass.

Dental concreting :

It is useful in case of preventing local failures generally in the stable slopes. Dental concreting may be substituted by the masonry. During both the shotcreting and dental concreting, blocking of the natural drainage system must be avoided.

Concrete ribs, beams and walls :

Concrete ribs and beams can serve as a resistant grid for the slope. Fully unstable slopes can be stabilized with gravity walls. Ribs , beams and walls can be combined with anchors. Continuous walls must include effective provisions for deep drainage of the rock-mass.

Drainage System :

a) Surface drainage arrangement

Surface drains can be ditched at the top of slope more or less parallel to it and also across the face vertical ditches spaced at an regular intervals to collect the water falling form the upper part for protecting the slope. Drainage conduits must be lined with softer lining materials and they must be provided with sufficient and safe evacuation devices.

b) Deep drainage system

To enhance the slope stability, the groundwater pressure must be reduced by facilitating the drainage system in a water-prone opencast mine. This is done by (i) drilling borehole from the surface (ii) drilling horizontal or inclined boreholes (from horizontal) into the slope face and (iii) excavating drainage galleries into the rockmass behind the slope. The deep drainage system is a very good support measure for big planar slides or mass instability of high wall slope. This system is useful when a continuous groundwater level surfaces the slope. Most soil landslides can be fully corrected by internal drainage only. In high-wall slopes internal drainage must be used in conjunction with other support measures (anchoring, walling, etc.) The deep drainage systems may be bone by –

1. **Vertical boreholes :** In case of vertical borehole method, boreholes are drilled from the surface at an predetermined intervals and up to the depth of the aquifer near to the level of floor of the pit. Then pumping is done for sucking water and major water are pumped out before the face is reached near to the borehole. The existing exploratory boreholes can also be used for this purpose. This method does not interfere the normal face production process .They are very effective if there is the presence of perched water level in the slope.

2. **Horizontal toe drains :** In case of horizontal or inclined hole (drilled from the face) method, the boreholes are drilled near to the toe of the slope and perforated pipes are inserted into them to guard against the caving of the rock mass and at the same time to allow the free flowing of water. Diameter of the hole may be 100mm and spacing between two holes is a round 6m. The water will flow by the gravity action and it must be canalized up to the mine sump by a proper drainage system .These drains are based horizontally with little bit upward inclination which is very effective to eliminate water pressure from the slope surface.

3. **Horizontal drainage adits :** In case of drainage galleries method, galleries are driven ahead of the bench and used for slope drainage by the action of gravity. Such type of galleries are immune to surface freezing and normal face production. They are also highly reliable and very effective . They are driven parallel to the slope and facilitate as a very effective measure for deep drainage.

Re-excavation/reshaping : Re-excavation / re-shaping of the disturbed bench may create a stable slope.

SPOIL DUMP SLOPE STABILITY

Problems of Spoil Dump Stability

The objective of stability analysis is to determine the factor of safety of any slip surface. The shape of potential slip surface and the mode of failure is governed basically by the behavior of the natural strata on which they are placed, nature of dumped materials, degree of compaction, dump height, slope angle, slope of the floor, seepage of water form dump to slope, changes in cohesion of interface material, changes of stresses, ground vibration by blasting, dynamic impact due to plying of dumpers , rollers etc.

Other factors of failure of stability of dump slope are due to the followings :

- Wash down of fines within the dump : This is because of deep erosion and formation of gullies. It is generally happens during the rainy season, causes voids in the dumps and lead to slope failure.
- Rotational slips : This happens because of decreasing the shear strength of material which is due to the decrease in frictional and cohesive components especially when seepage water pressurize dump material causing slope failure.
- Flow of mud (surface erosion) due to rapidly moving stream of water borne soil.
- Surface slips due to presence of dry and non-sticky materials in the overburden dump.
- Floor sliding due to presence of aquifer.
- Submergence of spoil dump.
- Back sapping by the interruption of mining slips.
- Creeping of the dumps due to clay slope, formation of artificial discontinuities like cutting of trenches, drains, or roads.
- Earthquakes

Spoil Dump Failure

Different type of spoil dump failure is as follows:

A) Cylindrical failure : It is semi-circular in cross-section. Under this category, the following are worth to mention :-

i) Deep seated failure : It occurs when there is the absence of any plane of weakness.

ii) Shallow cylindrical failure : It occurs when there are the presence of weak planes or other discontinuities.

B) Multiple block plane wedge failure : This type of failure is very common where dump is placed on a weak layers. This type of failure may be as follows :

i) Failure consisting of an active wedge and toe wedge.

ii) Wedge may be conceived to be consisting of an active block, a middle block and a passive block.

C) Plowslides : This type of failure include debris slides, mud flow, rocky mud flow, mud slide, earth flow, mud spate, etc.

With proper geo-technical engineering consideration and elaborate in-situ and laboratory testing of a foundation and dump material it is possible to economically design spoil dumps of desired height and slopes with adequate margin of safety. Choice of factor of safety in limit equilibrium methods of slope stability depends on the likely consequences of potential failure. High factor of safety demands where there is a risk of danger to persons or property and lower factor of safety may be taken into consideration for high dumps when located areas are not likely to endanger human lives and property. Table -2.3 shows the factor of safety commonly followed in mining industry abroad .

Various Precautionary Measures

The following precautionary measures to be taken:

- The mine management shall prepare – (a) a geological map of the area (in the scale of 1 in 10000 or other approved scale) sitting the location of dumps and neighboring land within 250 m of the boundaries of the dumps. (b) cross section of the strata (in a scale of less than 1 in 1250) underlying the dumps showing thickness, fault planes and geo-technical characteristics of strata including shear strength, angle of internal friction and cohesion of rock strata, etc. The mine management should also study the change of cohesion and angle of internal friction of dump material, change in cohesion and angle of internal friction of interface material between internal dump and mine floor, etc. (c) an accurate plan (on a scale of not less than 1 in 2500) showing all mine workings, previous land slides, springs, water courses and other natural and topographical features which might affect the security of the intended dumps. (d) plans and sections covering gradient of land, designed height, contours and boundaries of the dumps, position and nature of construction of any wall or other structure retaining or confining the dump, nature and location of the types of dump materials, the superficial area of the land etc.
- The angle of slope of any overburden dumps shall be less then the angle of repose of the material being dumped but in no case exceed 37.5° from the horizontal. The slope wall shall not be retained by the artificial means at an angle in excess of it's natural angle of repose.
- The dump height exceeding 30m shall be benched in such a manner that no bench exceeds 30m in height and the general slope shall in no case exceed 1 vertical to 1.5 horizontal. However with the increase in height of dump, seepage of a water, heavy ground vibration due to plying of HEMM and blasting, decreases in cohesion and angle of internal friction of dumped material , the slope angle to be decreased accordingly provided that the other factors remains unchanged and the same factor of safety are maintained.
- Toe of any dump shall not be permitted to approach any railway lines/public road, building or any other permanent structure not belonging to the owner of the mine, closer than a distance equal to the vertical height of the face.
- A suitable fence should be created around the toes of every dump so that no unauthorized person can approach any dump for under digging the same during collection of coal/stone coal.
- In case of internal dumps the coal ribs may be left on the floor which will give a good stability of the dumped material.
- Dumps shall be properly compacted as and when necessary.
- Sometimes judicious application of bolting, shotcreting etc. helps the stability of overburden dump slope.
- The slope may be stabilized using appropriate reshaping and earthwork measures including proper placement of soils and other materials.
- The dumping operation shall be carried out in such a way that the operations do not cause accumulation of water in, under or near the dump which may make the dump insecure. Build up of water on, inside or against up-hill toe of dump must be prevented and effectively drained. Water courses either natural or as a result of a torrential downpour shall not be adjacent in the dumps.

- Make the ring dump to stabilize old dumping wherever possible. This ring dumps shall also be covered by vegetation and shall have proper drainage arrangements and terracing.
- Steps to be taken to keep the top soil separated so that the same can be used to blanket/spray over the slope of the dumps and also on the top of them. Grass bed and plantation has to be made on the soils thereafter to consolidate the former.
- Toe wall of at least 1m thick and 2m height made by stone to be constructed all round the dumps to arrest the flow of soil. In addition to toe wall the stone wall at slopes at a vertical height of 5 – 10m have to be made to arrest soil at gradient.
- Piezometers are to be installed wherever required to determine the water pressure building within the dump.
- High percentage of soil and weak rocks should be kept separate to prevent mixing of soil dump and rock dump which will lead to sharp deterioration of shear strength value of total dump mass.
- The dumping and surrounding ground should be firm and solid to provide a stable foundation. Before dumping material quarry floor shall be cleared off any crushed coal/over-burden rock. With this measure the stability of floor shall be improved by thorough disrupting the potential failure plane through the weak interface between dump and foundation.
- A high degree of regular inspection of all the dumps by appropriate trained manpower is necessary. They shall monitor pegs on all the critical locations, detect fissures or other sign of instability, conditions of all the catchments areas, etc. It is worth to mention that any unstable surfaces or edges of dumps are designated by flags/pegs/posts/signs. For systematic observation modern stability instruments are essential. The mines manager or the nominated executive of the mine shall inspect all the active dumps once in a week, after a long periods of continuous rain or after a torrential downpour. Manager shall appoint a competent person to supervise the carrying of dumping operation, supervision of drainage system and its management and also for making and keeping the dumps in secured condition. All the closed dump, must be checked at an interval of not exceeding 6 months but in a frequent intervals during a long period of continuous rain or after a torrential downpour .

Table -2.2: Slope mass rating

SMR	Class of rock	Description of rock	Stability of highwall	Failures	Support requirement
81 – 100	I	Very good	Completely stable	None	None
61 – 80	II	Good	Stable	Some blocks	Occasional bolting
41 – 60	III	Normal	Partially stable	Some joints or many wedges	Systematic bolting/nets
21 – 40	IV	Bad	Unstable	Planar or big wedges	Corrective, systematic bolting / shotcrete / nets. Re-excavation
0 – 20	V	Very Bad	Completely unstable	Big planar or soil-like	Re-excavation

Interestingly it might be observed that the safe height of the spoil dumps with the presence and without the presence of coal rib are shown in the Table-2.4.

Table 2.3: Factor of safety commonly adopted for mining operations

Sl. No	U.K. Condition	FOS I	FOS II	CANADA Condition	FOS I	FOS II	USA Condition	FOS (Minimum)
1.	For slip surface along which the peak shear stress is used	1.5	1.25	Design based on peak shear strength parameters.	1.5	1.3	End of construction	1.3
2.	For a slip surfaces passing through a foundation stratum which is at its residual shear strength [slip circle should satisfy (1)]	1.35	1.15	Design based on residual shear strength parameter	1.3	1.2	Partial poor with steady seepage saturation	1.5
3.	For a slip surfaces passing along a deep vertical subsidence crack where no shear strength is created and which is filled up with water [slip surface should satisfy (1)]	1.35	1.15	Analysis based on inclusion of the predicted 100 years return period acceleration applied potential failure mass	1.2	1.1	Steady seepage from spillway or decant crest	1.5
4.	For a slip surfaces where both (2) and (3) apply	1.2	1.1	For horizontal sliding on the base of dyke in seismic area assuming shear strength of fine refuse in impoundment reduced to zero.	1.3	1.3	Earthquake (Case – II with loading)	1.0

FOS = factor of safety ,I = where there is a risk of danger involved for persons and property II = where there is no risk of danger involved for persons or property

Table 2.4: Height of dump w.r.t slope angle from horizontal

Internal dumps slope (degree)	Height of dump with coal rib (m)	Height of dump without coal rib (m)
35	80	67
32	90	80

Parameters considered for the above mentioned Table -2.4 : angle of internal friction of interface material : 37°, cohesion of interface material : 10 KN/m^2, inclination of floor : 3°, angle of internal friction of dump material : 33°, cohesion of dump material : 35 KN/m^2. There is no existence of water seepage force.

LENGTH OF FACE

Let an opencast mine has to produce Y Million Tonne ROM per annum .Let the height of bench, burden and spacing of the holes are h, b and s respectively.

The mine management decided to work the benches with single row blasting method.

Yield per hole = h.b.s.y tonnes

Where y =density of the ore

Ore production per week = $\frac{Yx10^6}{250}$ x 6 tonnes

Where number of net working days per annum = 250

So number of holes required per week = $\frac{\frac{Yx10^6 x6}{250}}{hxbxsxy}$

$= \frac{Yx10^6 x6}{250xhxbxsxy}$ ....(1)

So length of face required to accommodate above number of holes

$= \frac{Yx10^6 x6}{250xhxbxsxy}$ x spacing = $\frac{6x10^6 xY}{250xhxbxy}$ meter ... (2)

For production at shovel face another equivalent face length is also required for drilling.

So total face length = $\frac{2x6x10^6 xY}{250xhxbxy}$ meter . . . . . (3)

Recommended face length for grade control another face length as mentioned in the (3) will be required in the mine.

So a total face length is necessary = $\frac{2x2x6x10^6 xY}{250xhxbxy}$ meter

Let length of each bench is fixed to Z meter.

Then number of benches required = $\frac{24x10^6 xY}{Zx250xhxbxy}$

which should be rounded up to the nearest integer. Generally each bench should be provided with at least one shovel and one blast hole drill. More number of benches are to be worked out at a time to ensure full production and proper grade control .Width of the each bench should not be less than 5 times the width of a dumper.

When adequate face length will not be available ,the multi-row blasting (4 to 5 rows) with adequate delays should be preferred and hence number of benches will be reduced accordingly.

The main mine road is to be used for transportation of ore and development of lower horizon. For a large mine producing multi million Tonne/year, the ideal width of the road should be 4 to 5 times a dumper width or 16 - 17m since loaded dumpers and empty dumpers will be plying every minute in the main mine road. Gradient of haul road shall be 1 in 16. In the cliff side of the mine and along the running edge of benches safety bunds may be provided to prevent dumper's getting down the benches in night shift particularly during foggy weather.

So standard pattern of final slope-

Height of stripped bench-h meter

Inclination of bench- 45° to 70° depending upon the type of ore, pit slope stability etc.

Minimum width of bench floor is 10m for every 3rd bench from bottom in order to prevent rocks from falling down to the pit bottom, 5 m for others

Average final slope of pit = 40° to 45°

DESIGN OF HAUL ROADS

The design of the haul road should be such that with minimum cost it should have the high load bearing capacity and tally the designed traffic load. If the surface topography is gentle , heavy traffic is selected compare to the mines at hills since they have to laid on several severe grades. Water seepage during rainy season also causes severe land sliding problems. Adequate regular water sprinkling arrangements or salt zoning (spreading $CaCl_2$ or NaCl on the road surface and thereafter spraying of water will help to consolidate dust on the crust) system should be introduced for efficient suppression of dust and to stabilize the road surface. The design of haul road should tune up with the following:

- It should offer efficient transporting operation of mineral/ore/overburden.
- Selection of the site of the road should be such that it enables minimum cost of transportation of ore/coal/overburden and there is no weak ground or slope stability problems. The site of the road should be safe and has ease access to the mine working site.
- It should be sufficiently wide to avoid traffic congestion. No road shall be of a width less than 3 times the width of the largest vehicle plying on that road unless definite turn outs and waiting points are provided. One-way traffic is the best proposition for any surface mining operation. Pit design should incorporate allowances for reasonable future increase in equipment size.
- Very safe and less maintenance cost.
- Roads are to be maintained in good working condition.
- It should nicely balance the linkage among the crushing and screening plant, beneficiation plant, stock yards, waste dumps, etc.
- For better safety the optimum gradient of a haul road is around 7°, however the gradient may go up to 9° . When severe gradient comes to a question then for every 500 m, a patch of 50 m at a gradient of less than 1° to be provided for the purpose of safety after taking due consideration of drainage. Unless specially permitted by the Chief Inspector of DGMS, no road shall have a gradient steeper than 1 in 14 at any place. A gradient up to 1 in 10 may be permitted for designing the ramps.
- **Safe Stopping Distance**

In the haul road vertical and horizontal curves should be laid in such a manner that it will provide enough safe sight distance (not less than 30 m along the road). Sight and stopping distances are very important in both the curves. If it is not possible to ensure a visibility for a distance of 30 m, two roads (one for up and the other for down traffic) shall be provided.

The vehicle operators should have a clear and unobstructed visibility everywhere all along the road alignment, at level gradient or at the horizontal or vertical curves up to a distance within which he can safely stop his vehicle while traveling at a maximum allowable speed. Retarders will slow a unit to a constant speed but stopping ability is directly related to service brake efficiency. The stopping-sight distance has 2 distinct components.

One part consists of the distance traveled by the vehicle during the reaction time of the operator before applying the brake (a typical value being is 2.5 s) and the other is the braking distance which varies with the speed and weight of the vehicle. Table -2.5 and Table -2.6 depicts the stopping-sight distance for various speeds as per Indian Road Congress and the service brake maximum stopping distance of various categories of heavy vehicles at a speed of 32 km /hour respectively.

Sight distance is the distance up to which the vehicle operator gets a clear and unobstructed visibility and it shall never be less than the vehicle stopping distance.

Table – 2.5 Stopping-sight distance for various speeds as per Indian road congress.

Speed (km/h)	Perception and brake reaction		Braking		Safe stopping sight distance (m)
	Time (s)	Distance (m)	Coefficient of longitudinal friction	Distance (m)	
20	2.5	14	0.40	4	18
25	2.5	18	0.40	6	24
30	2.5	21	0.40	9	30
40	2.5	28	0.38	17	45
50	2.5	35	0.37	27	62
60	2.5	42	0.36	39	81
65	2.5	45	0.36	46	91
80	2.5	56	0.35	72	118
100	2.5	70	0.35	112	182

Table -2.6 Service brake maximum stopping distance of various categories of heavy vehicles at a speed of 32 km /hour

Category	Vehicle weight (Tonne)	Maximum stopping distance (m)
I	<45.4	18.30
II	45.4 to 90.8	27.40
III	90.8 to 181.6	30.10
IV	>181.6	53.30

- Road may be single lane one way traffic or two lane, two way traffic. Number of lanes depends upon the traffic density, normal safe distance between the trucks, truck speed, constraints of space, etc.
- Safe distance between the vehicle depends up on many factors e.g. gradient, speed of the vehicle, coefficient of adhesion, sensation of driver, etc.

Curves

In the curves additional allowance in width is to be added with both the curve and tangent to the curve, beside the standard of width(at least 3 times the width of the largest vehicle) as already mentioned. Avoid short radius curve .It will add high tyre cost, high maintenance cost , reduce productivity, etc.

The design of curves are governed by the factors like speed of the vehicle, condition of terrain, camber, road width and the size of the vehicles plying on the road. The degree of curve shall be small enough to avoid unproductive reduction in speed, it should fit the terrain in as much as economy and safety and should be wide enough to negotiate the minimum turning path capabilities of the vehicle. A vehicle while negotiating a horizontal curve is subjected to a centrifugal force, which has a tendency to make the vehicle slid outward away from the curve and thus exerts bending stresses on the vehicle frame and cause higher loads on the inside wheels and thus increases tyre wear. Vertical curves are required at locations where there is a change in gradient for providing smooth transition from one grade to another. The length of the vertical curves shall be adequate for a comfortable drive and to ensure adequate sight distance for the design speed is available. The rule-of-thumb for the road width or horizontal curve for the two-lane traffic is as follows :

$W = 2 (T + F_o + B_o + E) + L$

$L = E = (T + F_o + B_o) /2$

T= Track width of vehicle (centre to centre distance between the tyre), m

E = Extra width allowance due to the difficulty of driving on the curve, m.

F_o= Width of front overhang, m

B_o= Width of rear overhang, m

Sharp horizontal curves at the top and bottom of the ramps must be avoided. Sharp horizontal curve if necessary at or near the top of hill, shall be started well in advance of the vertical curve as the driver has difficulty in perceiving the curve during poor visibility, especially at night or in the foggy weather.

Recommendation for camber by the Indian Road Congress for highways which are usually adopted in mine haul roads also are given in Table -2.7 .

Table : 2.7 Recommendation for camber for haul roads

Sl . No	Type of surface	Recommended camber (%)	Indian road congress recommendation for highways (%)
1	Morum, gravel, stone including O.B. material giving rough surface	4.0	3.0
2	Overburden and other material resulting in smooth surface	3.0	2.5
3	Bituminous surface concrete road	2.5	2.0 to 2.5

Super-elevation

Super elevation (more than 0.2 m/m length) to be provided on curves to reduce the side friction between the tyres and the road surface. Super-elevation should be uniform throughout the tangent of the curve. Trucks should be driven at slow speed in the super elevation.

While negotiating curves the vehicle may be forced out ward by the centrifugal force ,if well precautionary measures are not taken. In a flat surface this centrifugal force is counteracted by the product of the vehicle weight and the side friction between the roadway and the tyres. In a particular velocity and radius the centrifugal force shall be equal or it may exceed the resisting force causing the vehicle skid sideways and topple. As a safeguard the roadways are elevated at the outer circumference of the curve which is called the super-elevation. It can be expressed as

$$e + f = \frac{v^2}{15R}$$

where e = The rate of super-elevation in m/m

f = Side friction factor

v = Vehicle speed in km/hr

R =Radius of curvature in m.

when f =0, no steering effort will be necessary to negotiate a curve. But when f is more than zero; the operator should steer up slope if the speed of the vehicle is slow or should steer down slope if the speed of the vehicle is very high to maintain the desired streamline path.

Excessive super-elevation should be avoided since trucks rounding a curve slips while in rain or when road is covered with ice. Over wetting can slide the vehicle inward and possibly overturn.

Table – 2.8 shows the recommended super elevation for mine haul roads in Australia.

Table – 2.8 Recommended super elevation for mine haul roads in Australia , m per m

Centre line radius of curve , m	Speed of vehicle, km/h				
	15	25	30	40	50
15	0.04	0.04	-	-	-
30	0.04	0.04	0.04	-	-
50	0.04	0.04	0.04	0.05	-
75	0.04	0.04	0.04	0.04	0.06
100	0.04	0.04	0.04	0.04	0.05
200	0.04	0.04	0.04	0.04	0.04
300	0.04	0.04	0.04	0.04	0.04

Culverts and drains

Culverts and drains are to be provided all along the haul roads. The factors which leads the design of the drainage systems depends upon the degree of rainfall, ground topography, catchments area, depth of the road base, etc.

During construction of the drain the following points are to be kept in mind-

(a) It should be constructed along the toe of the bench.

(b) It should not be constructed in the weak ground.

(c) The cross-sectional area of the drain should be enough to accommodate the expected amount of water.

(d) Gradient of the drain varies from 3% to 5% or more depending upon the hardness of the material.

(e) At the flat gradient drains are diverted to the natural drainage system or a pit sump through drains.

(f) 'V' drains which are very easy to construct should have a minimum slope angle of 2 : 1 and drain should be constructed on each side of the embankment when the haul road is constructed with fill materials.

Cross slope and culverts

If the slope of the drain is more than 8% , it is to be lined to control erosion and dissipating the velocity energy of water. The road surface should be crowned in the middle to facilitate water runoff. It is called the cross slope. In case of metallic road the cross slope of 1 in 50 is sufficient whereas in case of road made of crushed stone or by others the cross slope should be 1 in 25. A cross slope (1 in 50 to 1 in 25) is desirable to provide adequate drainage during heavy rain. On bench the cross slope should be inward direction whereas a haul road may have cross slope in one or on both the direction. If the metal pipe culverts are used , a cover of more than 60 cm and 90 cm over the pipe is essential for the weight of vehicle less than 100 t and more than 100 t respectively and a good lateral confinement is essential. To divert the water to the natural drainage system, they are installed at the low points beneath the haul roads, at all the intersections of the haul road, etc. The cross section of the galvanized or corrugated steel pipes should be sufficient to divert all the estimated water either in a drain or in a sump or beyond the toe of the slope of the spoil heap.

Construction of Haul Road

In most of the cases the haul road construction material is the mine product, crushed overburden, etc. But if a weak material is used for the road construction the surface will deteriorate at a faster rate or if the exposed competent rock/ore is used as a road, the presence of irregular and sharp edges of the competent rock will deteriorate the tyre and increase in fuel and maintenance cost. The construction of the road bed will be such that it enables the transfer of wheel loads over the foundation material in such a amount that it should not exceed the bearing capacity of the foundation material. The road bed may be divided into 3 courses which are surface course, base course and sub-base course.

During construction of road, the factors which are to be thought in depth are road bed thickness, widths, lengths, alignments, future possibilities of realignments, etc. The characteristics of the surface course should be of high adhesion, low rolling resistance coefficients, hard even surface, little dust, no penetration under load and should have low permeability. Concrete and asphalt are the best material for this purpose however finer crushed rock (+2 mm to -10 mm size), mill tailings, clean gravel etc. may also be used under compact stage. Sub-base course are generally made by compaction of layers of mine product or crushed overburden approximately 15 - 20 cm in thickness. It's width is more compare to the surface course. Compaction is done by the pressure of the roller. The material for the base course is generally-100 mm size crushed rock or high quality gravel which are to be compacted by the heavy roller. It would be very nice if large boulders, clay or silt is avoided for making a base course. The road must be graded to an even surface and a careful attention is to be paid for the contours of the pavement.

Generally a good road consists of 4 different layers which are as follows :-

i) Sub-grade: It is the foundation layer which supports all the loads coming on to the wearing surface. It may be made of compacted rock or soil or simply be the natural earth surface.

ii) Sub-base : The sub-base generally consists of rubbles which provides a good drainage, resist shrinkage, swelling, frost heave and distributes the load evenly. Generally it is provided over the weak sub-grade soil.

iii) Base: The base layer stables the roadway and distribute well the load offered by the vehicles on the wearing surface. It protects the working surface from any volume change, expansion, softening, displacement and frost penetration of sub-grade .

(v) *Wearing Surface*: The wearing surface may be made of asphalt concrete or crushed rock. It offers traction, resists abrasion and shear, reduces tractive resistance, transmits vehicle load on to the base in a well distributed manner and act as a seal against the penetration of surface water into the base.

The thickness of haul road crust depends on the following factors :

i) California bearing ratio (CBR) of sub-grade road construction material

ii) Wheel load

iii) Road adhesion co-efficient and rolling resistance offered by the road surface.

California bearing ratio method for determining the thickness of flexible pavement.

This approach characterizes the bearing capacity of given soil as a percentage of the bearing capacity of a standard crushed rock, and the ratio of the capacities being referred to as the CBR for the given soil. Empirical equivalent wheel load curves known as CBR curves relate the required pavement thickness and applied wheel load to the CBR value.

This method is extensively used by the highway engineers for evaluating the overall thickness and inter-layer thickness requirements for different sub-grades. The Table -2.9 shows the bearing capacities of various materials.

An example

Let a haul road pavement is to be designed for plying 120 Tonne capacity dumper and the pavement has to be laid over a sub-grade soil with a CBR value of 3. The soaked CBR for the construction materials available have been determined through field and laboratory tests and the results are as follows:-

Material	Soaked CBR
Crushed stone (compacted)	90%
Gravel (compacted)	60%
Mine spoil (compacted)	20%
Sub –grade soil (compacted)	3%

The 120 Tonne dumper has a wheel load of 80000 lbs. Interpolating this load on the CBR curves the following inferences obtained :

i) For sub-grade soil with CBR-3 the total fill thickness shall be 52 inches.

ii) Using mine spoil as sub-base material with a CBR-20 the required fill depth over the sub-base shall be 15 inches. Hence the sub-base thickness shall be 37 inches.

iii) Using gravel as base material with a CBR-60 the required fill depth over the base course shall be 7 inches. Therefore the thickness of the base course shall be 8 inches.

iv) Using crushed stone as surface material with a CBR-90 the required fill depth over the surface course shall be 7 inches.

It is necessary that the actual soaked CBR values for the construction materials proposed to be used and that of the sub-grade are determined through field and laboratory tests prior to finalization of design thickness.

The conventional method of haul road construction being costly, is rarely adopted in the opencast mines. The Russian method of utilizing the overburden material in haul road construction is perhaps ideally suited to Indian conditions. In this method sub-base is constructed out of run-off-mine sand-stone of 150 mm to 500 mm size. The immediate superjacent layer is the base course of 300 mm thick ROM sand stone of 50 mm to 160 mm size range. The top layer is made of 220 mm thick crushed hard stone layer of -50 mm size range.

Compaction

It is very important to emphasize the compaction requirements of the fill materials of the different sub-layers of the pavement to ensure stability of the final road surface. Compaction equipment shall preferably consists of heavy

smooth faced vibratory roller of above 15 Tonne capacity for compacting cohesion less materials and sheep foot roller for cohesive soils. If mines do not posses the said rollers, alternatively, loaded dumpers or large dozers could also be deployed.

Road Trafficability

Poor haul road trafficability and high vehicle rolling resistance are caused by the high ground water level within the road fill, presence of excessive fines in the surface and base courses along with high moisture content due to poor drainage facilities, insufficient thickness of road constructed over soft, marshy sub-grade soil, use of expansive type soil in the surface course, use of insufficient binder in the surface course and use of uniform size rather than well graded fill materials in the base and surface courses.

Excessive damages are created on the road due to the presence of high ground water. Water also causes high rolling resistance, poor bearing capacity of the road and also causes instability of the slope. Therefore road must have adequate drainage facilities, adequate camber (or cross fall) to ensure that there is no stagnation of water on the road surface at the time of precipitation. The rolling resistance comprises of 4 main components viz. (i) internal power train friction, (ii) tyre flexing under load (iii) tyre penetration and (iv) air resistance.

While the first 2 factors are nearly constant for a given vehicle, the other two factors may vary with the tyre condition and pressure. The penetration depends on the wheel load and road surface condition.

The rolling resistance can therefore be minimized by constructing and maintaining a well compacted haul road with smooth surface.

The materials for haul road surface which will ensure maximum safety and operational efficiency are -

i) Roll-crete - It is the roller compacted cement concrete with a very low water cement ratio.

ii) Asphaltic concrete - It is the mixture of stone chips with the asphalt.

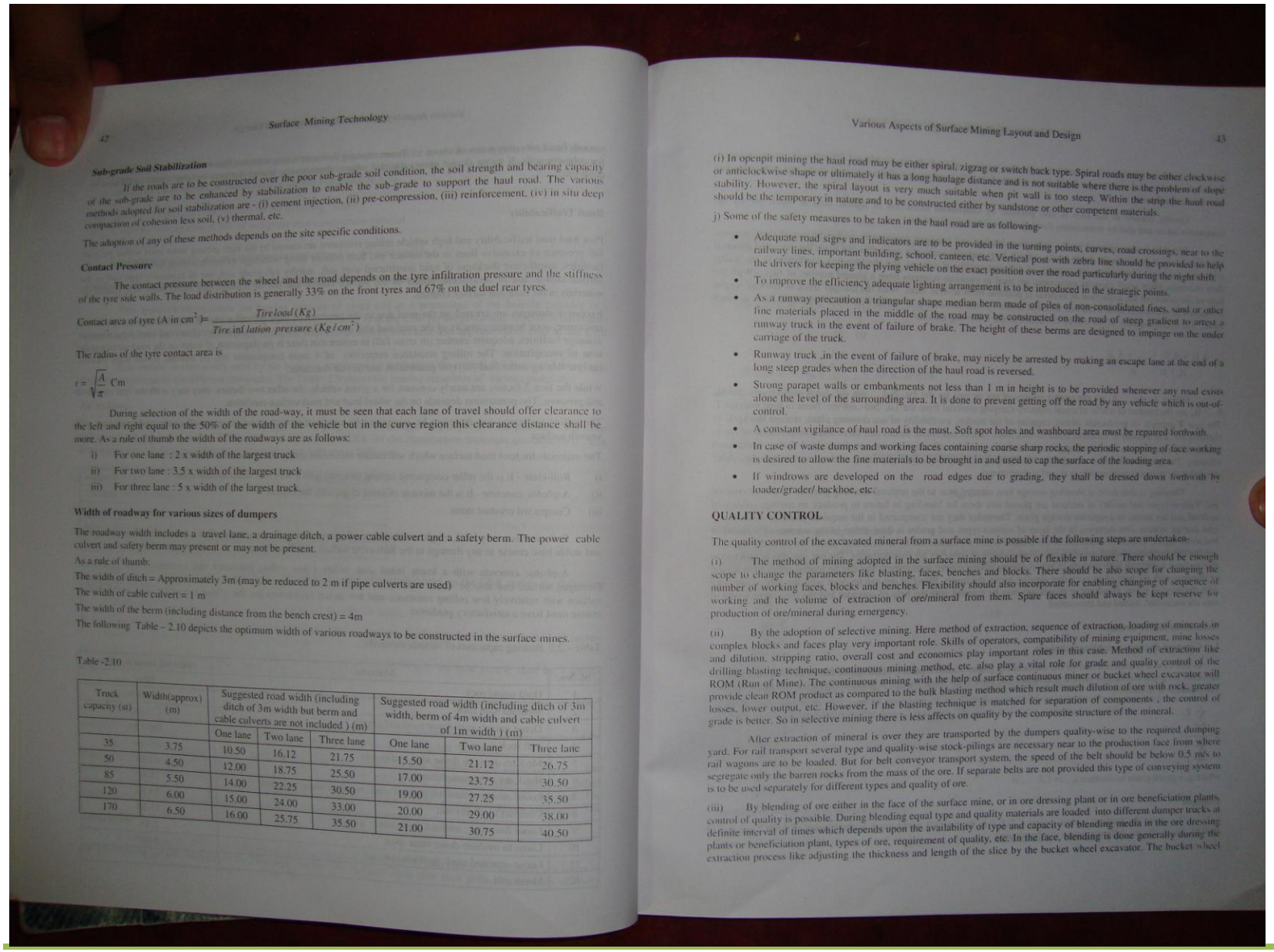
iii) Compacted crushed stone

Roll-crete is a very hard, tough and durable surface with low rolling resistance. But it has to be laid on a very firm and stable base course as any damage to the Roll-crete surface is difficult to repair. It is also very expensive.

Asphaltic concrete with a lower initial cost, offers a low rolling resistance and minimizes dust generation. Damaged surface can also be easily repaired in this material. Compacted crushed stone provides a very good stable surface with relatively low rolling resistance and low initial investment but the granular material used in the surface course must have a satisfactory gradation.

Table – 2.9 Bearing capacities of various materials.

Sl. No.	Material	Bearing capacity (PSI)
1	Hard sound rock	800+
2	Medium hard rock	400-600
3	Dense gravel ,very dense sand and gravel	120-160
4	Soft rock	100-120
5	Medium dense to dense sand and gravel	80-100
6	Hard ,dry over consolidated clay	60-80
7	Loose coarse to medium grained sand , medium dense fine sand	30-60
8	Compact sand clay soil	30-40
9	Loose fine sand , medium dense sand –silt soil	15-30
10	Loose to medium dense mine spoil	5-15
11	Loose saturated sand , medium soft clay	5-10
12	Marsh soil	0-5



Sub-grade Soil Stabilization

If the roads are to be constructed over the poor sub-grade soil condition, the soil strength and bearing capacity of the sub-grade are to be enhanced by stabilization to enable the sub-grade to support the haul road. The various methods adopted for soil stabilization are - (i) cement injection, (ii) pre-compression, (iii) reinforcement, (iv) in situ deep compaction of cohesion less soil, (v) thermal, etc.

The adoption of any of these methods depends on the site specific conditions.

Contact Pressure

The contact pressure between the wheel and the road depends on the tyre infiltration pressure and the stiffness of the tyre side walls. The load distribution is generally 33% on the front tyres and 67% on the duel rear tyres.

$$\text{Contact area of tyre (A in cm}^2\text{)} = \frac{\text{Tire load (Kg)}}{\text{Tire inflation pressure (Kg/cm}^2\text{)}}$$

The radius of the tyre contact area is

$$r = \sqrt{\frac{A}{\pi}} \text{ Cm}$$

During selection of the width of the road-way, it must be seen that each lane of travel should offer clearance to the left and right equal to the 50% of the width of the vehicle but in the curve region this clearance distance shall be more. As a rule of thumb the width of the roadways are as follows:

i) For one lane : 2 x width of the largest truck
ii) For two lane : 3.5 x width of the largest truck
iii) For three lane : 5 x width of the largest truck.

Width of roadway for various sizes of dumpers

The roadway width includes a travel lane, a drainage ditch, a power cable culvert and a safety berm. The power cable culvert and safety berm may present or may not be present.

As a rule of thumb:

The width of ditch = Approximately 3m (may be reduced to 2 m if pipe culverts are used)

The width of cable culvert = 1 m

The width of the berm (including distance from the bench crest) = 4m

The following Table – 2.10 depicts the optimum width of various roadways to be constructed in the surface mines.

Table -2.10

Truck capacity (st)	Width(approx) (m)	Suggested road width (including ditch of 3m width but berm and cable culverts are not included) (m)			Suggested road width (including ditch of 3m width, berm of 4m width and cable culvert of 1m width) (m)		
		One lane	Two lane	Three lane	One lane	Two lane	Three lane
35	3.75	10.50	16.12	21.75	15.50	21.12	26.75
50	4.50	12.00	18.75	25.50	17.00	23.75	30.50
85	5.50	14.00	22.25	30.50	19.00	27.25	35.50
120	6.00	15.00	24.00	33.00	20.00	29.00	38.00
170	6.50	16.00	25.75	35.50	21.00	30.75	40.50

(i) In openpit mining the haul road may be either spiral, zigzag or switch back type. Spiral roads may be either clockwise or anticlockwise shape or ultimately it has a long haulage distance and is not suitable where there is the problem of slope stability. However, the spiral layout is very much suitable when pit wall is too steep. Within the strip the haul road should be the temporary in nature and to be constructed either by sandstone or other competent materials.

j) Some of the safety measures to be taken in the haul road are as following-

- Adequate road signs and indicators are to be provided in the turning points, curves, road crossings, near to the railway lines, important building, school, canteen, etc. Vertical post with zebra line should be provided to help the drivers for keeping the plying vehicle on the exact position over the road particularly during the night shift
- To improve the efficiency adequate lighting arrangement is to be introduced in the strategic points.
- As a runway precaution a triangular shape median berm made of piles of non-consolidated fines, sand or other fine materials placed in the middle of the road may be constructed on the road of steep gradient to arrest a runway truck in the event of failure of brake. The height of these berms are designed to impinge on the under carriage of the truck.
- Runway truck ,in the event of failure of brake, may nicely be arrested by making an escape lane at the end of a long steep grades when the direction of the haul road is reversed.
- Strong parapet walls or embankments not less than 1 m in height is to be provided whenever any road exists alone the level of the surrounding area. It is done to prevent getting off the road by any vehicle which is out-of-control.
- A constant vigilance of haul road is the must. Soft spot holes and washboard area must be repaired forthwith.
- In case of waste dumps and working faces containing coarse sharp rocks, the periodic stopping of face working is desired to allow the fine materials to be brought in and used to cap the surface of the loading area.
- If windrows are developed on the road edges due to grading, they shall be dressed down forthwith by loader/grader/ backhoe, etc.

QUALITY CONTROL

The quality control of the excavated mineral from a surface mine is possible if the following steps are undertaken-

(i) The method of mining adopted in the surface mining should be of flexible in nature. There should be enough scope to change the parameters like blasting, faces, benches and blocks. There should be also scope for changing the number of working faces, blocks and benches. Flexibility should also incorporate for enabling changing of sequence of working and the volume of extraction of ore/mineral from them. Spare faces should always be kept reserve for production of ore/mineral during emergency.

(ii) By the adoption of selective mining. Here method of extraction, sequence of extraction, loading of minerals in complex blocks and faces play very important role. Skills of operators, compatibility of mining equipment, mine losses and dilution, stripping ratio, overall cost and economics play important roles in this case. Method of extraction like drilling blasting technique, continuous mining method, etc. also play a vital role for grade and quality control of the ROM (Run of Mine). The continuous mining with the help of surface continuous miner or bucket wheel excavator will provide clean ROM product as compared to the bulk blasting method which result much dilution of ore with rock, greater losses, lower output, etc. However, if the blasting technique is matched for separation of components , the control of grade is better. So in selective mining there is less affects on quality by the composite structure of the mineral.

After extraction of mineral is over they are transported by the dumpers quality-wise to the required dumping yard. For rail transport several type and quality-wise stock-pilings are necessary near to the production face from where rail wagons are to be loaded. But for belt conveyor transport system, the speed of the belt should be below 0.5 m/s to segregate only the barren rocks from the mass of the ore. If separate belts are not provided this type of conveying system is to be used separately for different types and quality of ore.

(iii) By blending of ore either in the face of the surface mine, or in ore dressing plant or in ore beneficiation plants, control of quality is possible. During blending equal type and quality materials are loaded into different dumper trucks at definite interval of times which depends upon the availability of type and capacity of blending media in the ore dressing plants or beneficiation plant, types of ore, requirement of quality, etc. In the face, blending is done generally during the extraction process like adjusting the thickness and length of the slice by the bucket wheel excavator. The bucket wheel

By:- Vikas shrimali & Vipul bansal

excavator loads simultaneously several heterogeneous partings or load alternately blasted off different quality minerals from individual section of a face or firstly form intermediate piling and coning of the extracted coal / ores before loading into the dumper trucks.

Quality control in the face is also possible by continuous extraction of lignite, coal or weak ore by the continuous miner and also by maintaining flowing of consistent quality of ore.

Blending is also affected by the extraction, loading and transport equipment and by the use of their systems. Blending factor is greater in the dump trucks, when a large capacity excavator excavates and loads the extracted minerals with more number of heterogeneous partings of different quality of mineral. But the blending coefficient in the traffic flow increases when small capacity excavators are used to excavate minerals from a greater number of faces. With the help of the conveyor the collection of mineral is possible from several faces. With the dumper truck or railway wagon collection of ore/coal is done from one face but in this system monitoring of traffic flow from individual faces is much easier. The average quality of ore can be calculated in the following manner-

$$A=\frac{A_1B_1+A_2B_2+\ldots\ldots+A_nB_n}{B_1+B_2+\ldots B_n}$$

Where A_1, A_2 ... A_n are the quality indices of ore or coal for the 1st, 2nd ....nth etc. faces respectively and B_1, B_2,... B_n are the number of traffic of equal capacity carrying ore/coal from the 1st, 2nd ..nth etc. faces in time interval T. The time **T** depends on production of ore, capacity of the bunker, availability of ore dressing plant, storage stockpile of blending etc. The amount of different grade of ores from various individual blocks and faces and transportation of ore in calculated proportions to the blending storage stockpile or in beneficiation plants are determined during operational planning .The quality of ROM is ensured practically by operation and control of mining methods and techniques. After doing that the output requirement by a shovel is determined.

Blending is also done at blending storage bins situated near to the ore/coal dressing plant to blend the crushed ore. Various types and quality of ore/coal are placed into them for blending in future to produce an average quality of ore/coal and are stored in a separate storage place. Thereafter they are transported in the required place. In stock-dump type storage system, first dumping in the layer of various types and grades is done either in the manner of inclined or horizontal layers. Thereafter extraction of the dumped layers are carried out by an excavator across the different layers of ore and they are either loaded onto dumper, railway wagon or conveyor for transportation. The ore can also be stocked in thin inclined layer cone with the help of belt conveyor having tripper arrangement and can be loaded either onto wagon or dumpers with the help of an excavator digging across the layers. Piling of ore can be done with the help of spreader with discharging boom. Hundreds of thin layers may be dumped with the help of spreader and the transverse layers of ores are excavated, loaded and transported.

Assessment of Ore Grades

1. Constant distance weighting technique

In this case it is assumed that all the surrounding grades around a point (whose grade has to be estimated) has an influence on grade estimation. Here it is assumed that there is the linear change in grade between the two known grades. The equation is as follows :

$$g=\frac{\sum_{i=1}^{n}\frac{g_i}{z_i}}{\sum_{i=1}^{n}\frac{1}{z_i}} \qquad \ldots (1)$$

where g_i = given grade at distance z_i, i= 1,2, ...n

Example

To assess the grade at various points C,D,E,F in the line AB with known grades (grade of A =0.7 and that of B=0.6) (Fig.2.10) ,the following steps can be adopted :

i)Grade at **C** , $g_C=\dfrac{\frac{0.7}{5}+\frac{0.6}{20}}{\frac{1}{5}+\frac{1}{20}}=0.68$ or 68%

ii) Grade at **D** , $g_D=\dfrac{\frac{0.7}{10}+\frac{0.6}{15}}{\frac{1}{10}+\frac{1}{15}}=0.66$ or 66%

iii)Grade at **E** , $g_E=\dfrac{\frac{0.7}{15}+\frac{0.6}{10}}{\frac{1}{15}+\frac{1}{10}}=0.64$ or 64 %

iv)Grade at **F** , $g_F=\dfrac{\frac{0.7}{20}+\frac{0.6}{5}}{\frac{1}{20}+\frac{1}{5}}=0.62$ or 62%

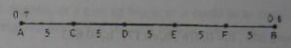


Fig. 2.10

2. Inverse distance weighting technique

In this case straight lines are drawn to a point (where grade is to be determined) from the points of surround different grades. Their distances are measured and thereafter the values are put in the following formula:

$$g=\frac{\sum_{i=1}^{n}\frac{g_i}{z_i}}{\sum_{i=1}^{n}\frac{1}{z_i}} \qquad \ldots (2)$$

3. Inverse distance squared weighting technique

Since the grade of block shall be more similar to nearer points than those situated in the far away the weighting with distance should be increased by the power of z_i . Therefore in this technique the dependence varies inversely with the power of the distance rather than a linear way. The formula for this technique is as follows :

$$g=\frac{\sum_{i=1}^{n}\frac{g_i}{z^{p}_{i}}}{\sum_{i=1}^{n}\frac{1}{z^{p}_{i}}} \qquad \ldots (3)$$

The value of p=1 in case of linear dependence

P=2 in case of squared dependence, etc.

However widely accepted value of p is 2. Therefore the equation will become :

$$g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^2}}{\sum_{i=1}^{n} \frac{1}{d_i^2}} \quad \ldots (4)$$

Let the following figure be considered for this method. The grade (g) at the point p shall be

$$g = \frac{\frac{g_1}{25} + \frac{g_2}{16} + \frac{g_3}{9} + \frac{g_4}{4} + \frac{g_5}{1}}{\frac{1}{25} + \frac{1}{16} + \frac{1}{9} + \frac{1}{4} + \frac{1}{1}}$$

$$= 0.027\, g_1 + 0.043 g_2 + 0.076 g_3 + 0.171 g_4 + 0.683 g_5 \quad \ldots (5)$$

The above equation No -5 can be written as

$$g = b_1 g_1 + b_2 g_2 + b_3 g_3 + b_4 g_4 + b_5 g_5 \quad \ldots (6)$$

$$b_i = \frac{\frac{1}{d_i^2}}{\sum_{i=1}^{n} \frac{1}{d_i^2}}$$

The coefficient : b_1=0.027, b_2=0.043 , b_3= 0.076 , b_4=0.171, b_5=0.683

The sum of the coefficient

$$\sum_{i=1}^{n} b_i = 0.027+0.043+0.076+0.0171+0.683 = 1$$

Therefore $0 \le b_i \le 1$

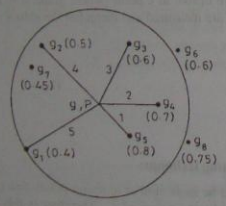


Fig. 2.11

If the value of g_1=0.4 , g_2=0.5, g_3=0.6 , g_4=0.7 , g_5=0.8

The value of g i.e. grade of ore at p will be

g = 0.027x 0.4 +0.043x0.5+0.076x0.6+0.171x0.7+0.683x0.8 =0.744 or 74.4%

In case of inverse distance weighting technique the value of g i.e. the grade of ore at the point p (fig. 2.11) will be :

$$g = \frac{\frac{0.4}{5} + \frac{0.5}{4} + \frac{0.6}{3} + \frac{0.7}{2} + \frac{0.8}{1}}{\frac{1}{5} + \frac{1}{4} + \frac{1}{3} + \frac{1}{2} + \frac{1}{1}} = 0.681 \text{ or } 68.1\%$$

4. Statistical method

Kriging

Kriging is an estimation method that gives the minimum variance unbiased linear estimates of point values or of block averages. The accuracy of the estimates depends on several factors which are as follows :

- The number of samples and the quality of the data at each point.
- The positions of the samples within the deposit. Evenly spaced samples achieve better coverage and thus give more information about the deposit than clustered samples .
- The distance between the samples and the point or block to be estimated.
- The spatial continuity of the variable under consideration. It is easier to estimate the value of a fairly regular variable than one which fluctuates wildly.

Let us have N data values $z(x_1)$, ... $z(x_n)$at our disposal and let us estimate a linear function of the variable Z(x). For example : estimate its value at a particular point, $Z(x_o)$,or its average over a certain region. The quantity to be estimated by:

$$z_v = \frac{1}{V}\int_V z(x)dx$$

The volume V could be the whole deposit, or a mining block, or it could be as small as a single point in the case of point estimation. To estimate Z(V), a weighted average of the data may be considered:

$$z\hat{v} = \sum \lambda_i z(x_i)$$

where λ_i are the weighting factors. By convention, the star will be used to denote the estimated value as opposed to the real but unknown value. The problem is to choose the weighting factors in the best way. The regional variable will be:

$$Z\hat{v} = \sum \lambda_i Z(x_i)$$

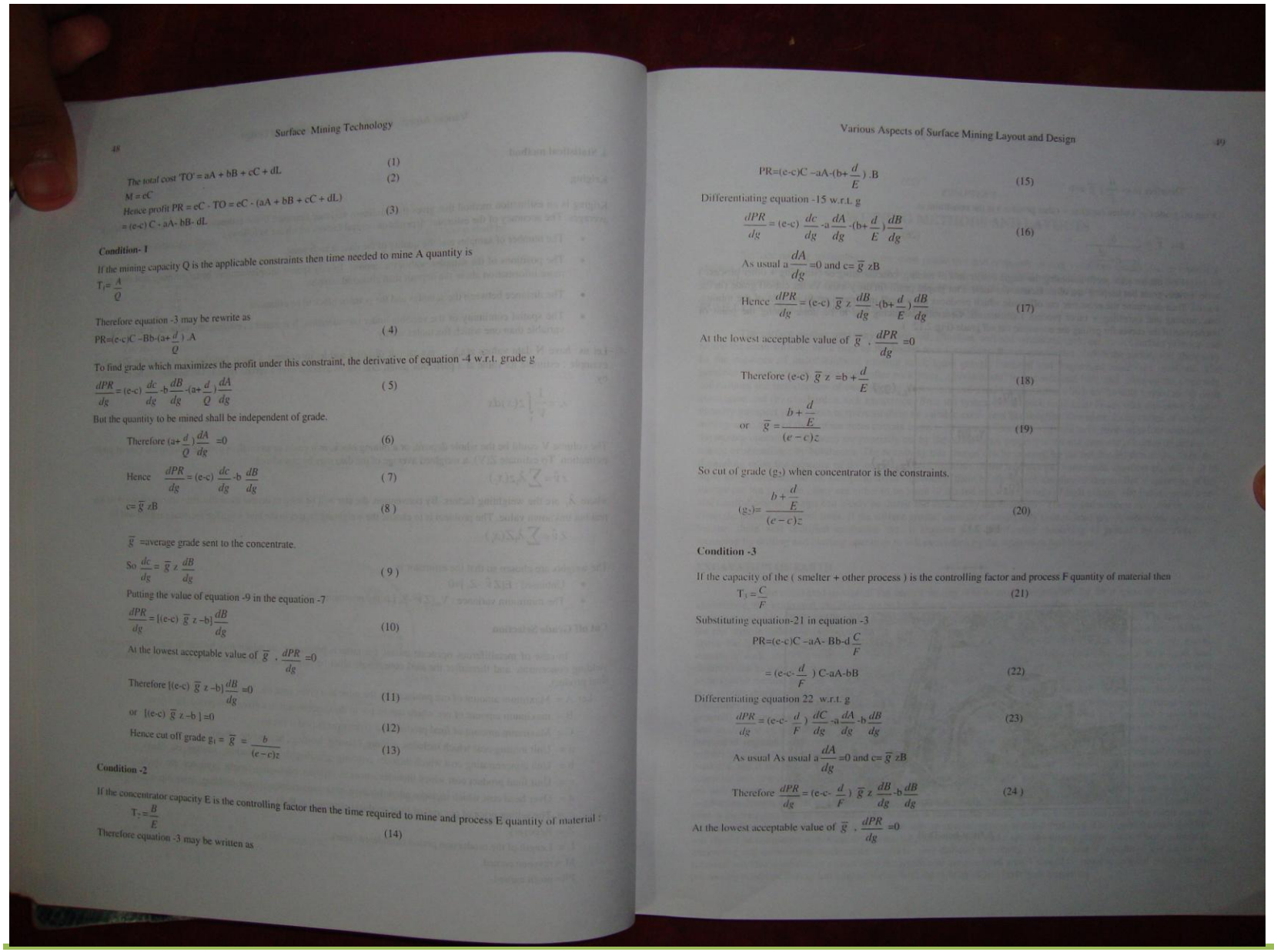
The weights are chosen so that the estimator is:

- Unbiased : $E[Z\hat{v} - Z_v] = 0$
- The minimum variance : $V_{ar}[Z\hat{v} - Z_v]$ is the minimum. The variance is called the kriging variance.

Cut off Grade Selection

In case of metalliferous opencast mines, the mine is producing ore which may be sent to the concentrator for yielding concentrate and thereafter the said concentrate shall be sent to the smelter and other process for yielding the final product.

Let A = Maximum amount of ore produced by the mine in a given time i.e., t/year.
B = maximum amount of ore which can be put in the concentrator in a given time period i.e. t /year.
C = Maximum amount of final product in a given time period (t / year)
a = Unit mining cost which includes, drilling, blasting, loading , hauling etc. (Rs/t)
b = Unit concentrating cost which includes crushing ,grinding, floating, leaching, haulage, etc. (Rs/t).
c = Unit final product cost which includes smelting, refining, packaging, freight, insurance, etc. (Rs/t).
d = Over head cost which includes administrative, rent, maintenance of road, building, township, etc. (Rs/year)
e = The selling price of per unit of final product (expressed in Rs/t)
Z = Recovery
L = Length of the production period considered (year).
M = revenue earned.
PR= profit earned.



The total cost 'TO' = aA + bB + cC + dL (1)

M = eC (2)

Hence profit PR = eC - TO = eC - (aA + bB + cC + dL)

= (e-c) C - aA- bB- dL (3)

Condition- I

If the mining capacity Q is the applicable constraints then time needed to mine A quantity is

$T_1 = \frac{A}{Q}$

Therefore equation -3 may be rewrite as

$PR=(e-c)C - Bb-(a+\frac{d}{Q}) .A$ (4)

To find grade which maximizes the profit under this constraint, the derivative of equation -4 w.r.t. grade g

$\frac{dPR}{dg} = (e-c) \frac{dc}{dg} -b\frac{dB}{dg} -(a+\frac{d}{Q})\frac{dA}{dg}$ (5)

But the quantity to be mined shall be independent of grade.

Therefore $(a+\frac{d}{Q})\frac{dA}{dg} =0$ (6)

Hence $\frac{dPR}{dg} = (e-c) \frac{dc}{dg} -b \frac{dB}{dg}$ (7)

$c= \bar{g}\ zB$ (8)

$\bar{g}$ =average grade sent to the concentrate.

So $\frac{dc}{dg} = \bar{g}\ z \frac{dB}{dg}$ (9)

Putting the value of equation -9 in the equation -7

$\frac{dPR}{dg} = [(e-c)\ \bar{g}\ z -b]\frac{dB}{dg}$ (10)

At the lowest acceptable value of $\bar{g}$, $\frac{dPR}{dg} =0$

Therefore $[(e-c)\ \bar{g}\ z -b]\frac{dB}{dg} =0$ (11)

or $[(e-c)\ \bar{g}\ z -b\ ] =0$ (12)

Hence cut off grade $g_1 = \bar{g} = \frac{b}{(e-c)z}$ (13)

Condition -2

If the concentrator capacity E is the controlling factor then the time required to mine and process E quantity of material :

$T_2 = \frac{B}{E}$ (14)

Therefore equation -3 may be written as

$PR=(e-c)C - aA-(b+\frac{d}{E}) .B$ (15)

Differentiating equation -15 w.r.t. g

$\frac{dPR}{dg} = (e-c) \frac{dc}{dg} -a\frac{dA}{dg} -(b+\frac{d}{E})\frac{dB}{dg}$ (16)

As usual $a\frac{dA}{dg} =0$ and $c= \bar{g}\ zB$

Hence $\frac{dPR}{dg} = (e-c)\ \bar{g}\ z \frac{dB}{dg} -(b+\frac{d}{E})\frac{dB}{dg}$ (17)

At the lowest acceptable value of $\bar{g}$, $\frac{dPR}{dg} =0$

Therefore $(e-c)\ \bar{g}\ z = b+\frac{d}{E}$ (18)

or $\bar{g} = \frac{b+\frac{d}{E}}{(e-c)z}$ (19)

So cut of grade (g_2) when concentrator is the constraints.

$(g_2) = \frac{b+\frac{d}{E}}{(e-c)z}$ (20)

Condition -3

If the capacity of the (smelter + other process) is the controlling factor and process F quantity of material then

$T_3 = \frac{C}{F}$ (21)

Substituting equation-21 in equation -3

$PR=(e-c)C - aA- Bb-d\frac{C}{F}$

$= (e-c-\frac{d}{F}) C-aA-bB$ (22)

Differentiating equation 22 w.r.t. g

$\frac{dPR}{dg} = (e-c-\frac{d}{F}) \frac{dC}{dg} -a\frac{dA}{dg} -b\frac{dB}{dg}$ (23)

As usual As usual $a\frac{dA}{dg} =0$ and $c= \bar{g}\ zB$

Therefore $\frac{dPR}{dg} = (e-c-\frac{d}{F}) \bar{g}\ z \frac{dB}{dg} -b\frac{dB}{dg}$ (24)

At the lowest acceptable value of $\bar{g}$, $\frac{dPR}{dg} =0$

Therefore $(e-c-\frac{d}{F})\ \bar{g}\ z=b$ (25)

Or cut off grade (g_3) when (smelter + other process) is the constraint is

$$g_3,\ \bar{g} = \frac{b}{(e-c-\frac{d}{F})z} \quad (26)$$

Find out the total profit assuming the single constraints of mining, concentrating or (smelting + other process) under various grade but keeping the other factors the same. Plot graphs profit (in the y-axis) Vs the cut-off grade (in the x-axis). Then determine the economic cut off grade which produces the overall maximum profit in light of the mining, concentrating and (smelting + other process) constraints. Generally balancing has to be done among the point of intersection of the curves for getting the economic cut off grade (Fig-2.12).

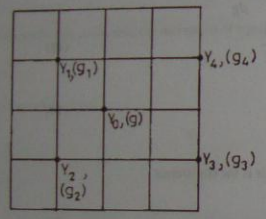
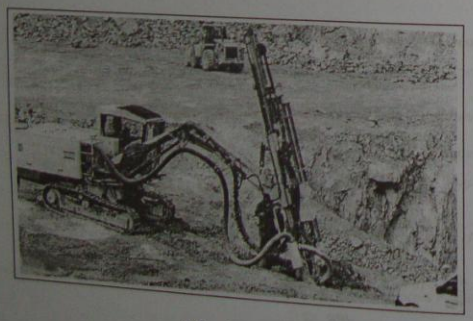


Fig. 2.12



A Blast-hole Drill

CHAPTER 3

WINNING METHODS AND LAYOUTS

Surface mining technology is the most productive and economic mining system to extract coal or mineral in shallow depth. A well designed pit layout is the most vital option to yield more production and productivity taking all due consideration of safety measures, environmental and ecological balance including protection and also economy.

BUSH CLEANING, GRUBBING AND SCARIFICATION

The surface mining area may contain soil, alluvium, soft and weathered rock, etc. on it's surface . On top and below the earth surface, there may be the harder rock formations. But in some cases , below the surface soil ,there may be the presence of unconsolidated running sand, loose gravel, fractured and fragmented loose rock mass, etc. The presence of surface soil, loose and softer rock mass, alluvium, etc. in one side and hard rock mass on the other side necessitates the introduction of two types of excavators and excavation systems which are broadly termed as (a) earth excavation and (b) overburden rock excavation. Both the system extract rock mass from in situ rock and either dispose them by transport sub-system or overcast them by suitable excavators like dragline, shovel etc. Except some of the barren mining area, most of the surface areas contain bushes, forests etc. which are required to be removed before commencing the mining operation. The trees are generally cut by the power saws, properly sized and disposed off either by the trucks, tractor or sometimes by bulldozers. The remaining tree stumps can be grubbed by the bulldozers, shovel, draglines, etc. depending upon the capacity of the equipment and the sizes of the tree stumps. Sometimes scarifiers are also used to loose the ground around the stumps to assist uprooting them by the shovel, dragline or dozers. But if uprooting of the stumps are not possible , they are either to be burnt or blasted out with the help of light charge. The bushes, small trees and small diameter stumps can nicely be dozed and cleaned by the bulldozer. Tractor and scrapers may also be used to clean the bushes and small trees. If the surface ground consists of moderately consolidated gravel, weathered rock, loose laterite, shale, soft stratified sandstone, etc., it requires prior scarification (loosening of ground) by the rippers or loosening by drilling and blasting operation to suit excavation by the scrapers or bulldozers.

EXCAVATION OF EARTH

Since the excavated portion of the surface mining is to be reclaimed properly for the purpose of agriculture or aforestation, the excavated soil layers are to be dumped in a separate place one after another in a manner matching the surface topography. The top layer of the soil contains most of the organic matter like flora, fauna, etc. The layer below the top layer is called the subsoil which contains more clay, alumina, iron, etc. whereas the bottom most layer which is above the hard rock formation contains mainly loose or weathered rocks. Since the earth materials (soil, gravel, weathered rock, stratified soft sandstone at the surface, etc) are very soft, scraper, bulldozer (for a very small haul distance) can be used for excavation directly and for their disposal too. Due care is to be taken to excavate the earth with some special method and they are to be transported and stacked layer after layer according to the chronological sequence in a proper dumping place. If the ground is moderately consolidated or stiff in nature, prior scarification or even sometimes drilling and blasting operation is necessary to suit excavation by the scrapers or bulldozers. Dumping place is generally selected based on the availability of barren land, land which is not to be excavated in future, and it must be as near to the site of extraction to minimize the cost. If low lying barren areas are available they may be used for the purpose of vegetation after proper reclamation with the help of waste available due to surface mining. As far as possible number of overburden earth bench should be one. If the total thickness of the earth is more and drilling and blasting is practiced and if the distance of waste dump is more than 500 m , front-end-loader and truck combinations is the most economic and best compare to a scraper. However it would be better if the top most soil layer and subsoil are separated out and dumped near to the waste dump for the purpose of future use in the best manner after the reclamation. The overburden dumps, mineral stockyards, etc should be located on the barren land, away from the fertile land or forest land even if the transport cost of overburden and mineral is more. If the mining work is to be carried out in the forest area an equivalent compensatory forestation is to be done over an equivalent area of non-forest land. The required removal of top soil should be completed within the dry season otherwise rain will make soil into sloppy plastic muck where moving of excavating and transporting machineries would be very difficult and costly. It will be best if individual soil layers are removed and stored separately so that after reclamation of the extracted area it should be equal to original quality to it's pre-mining condition having less contamination with the mixture of sub-layer rock fragments.

BOX CUT

Opening up of open pits is done by an opening cut for the development of first working bench. The opening cut is called the box cut and slope of suitable gradient (steepest is advisable both technical and economical point of view for transports, holding space and minimizing the cost of excavation for deep pits) is necessary. The box cut may be internal (located on the bench floor on one side of the pit). It saves cost of excavation and suitable for deep pits since number of benches in this case is more. Remember that excavation by external means is extremely large and expensive) and external box cut is suitable for shallow pits. Box cut (Fig 3.1) is excavated initially down to the floor level of the first bench from the surface. Thereafter an opening level trench is extended from this opening cut to form the first bench. The opening trench is narrow keeping due regards of the turning of the machineries used for excavation and extends along or across the quarriable limit depending on the type of the deposit. When the first bench is sufficiently advanced, the box cut is oriented and extended to the next lower bench keeping due regard of the sufficient amount of rooms for the approach road to the top (1st) bench and for opening trench for the 2nd bench. This way a number of working benches are developed and the width of the box cut should be sufficient enough to diversify the approach road to all the benches. If number of benches are developed from one opening cut, the cut should be started enough away from the pit limit so that bottom bench can be reached at the desired slope of the pit. This type of opening cut may be very long and may be curved depending upon the shape and extent of the deposit. For opening up in hilly deposit, a central trench cut is given across the top level for the first bench or from one side in the same contour level forming a length of face which will give the required production rate.

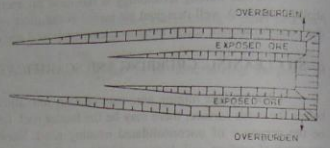


Fig. 3.1 : Excavation of Box Cut.

Location of Box Cut

The location of the box cut depends upon the following factors :

- Minimum cost of haulage within the open-pit and outside the pit to the desired place e.g. preparation plant, sidings, etc. or mineral and overburden dump. However, if the place of mineral siding and overburden dump is far apart, separate opening cuts one for mineral and the other one for overburden is to be formed.
- If shifting of box cut is not necessary , it is to be located at the boundary of the mineral deposit/property. In dipping deposit, this boundary should be the mineral outcrops or where ratio of overburden: ore is least, location of the box cut should be in the middle of the boundary of the reserve for minimizing the haulage cost.
- In horizontal deposit, the box cut can be located in the middle of any boundary depending on the location of the place of destination of the overburden and mineral disposal. The opening trench in continuation of the box cut offers two working faces of adequately long for the desired production on each level.
- The site of the box cut should be stable and free from geological disturbances.
- The site should be at the rise side to guard against the flooding of the mine.
- The site should be selected where the construction of approach road is very convenient.
- The site should be selected preferably where the deposit has high grade mineral to compensate the development cost.
- Location of the box cut should be such that it serves the purpose of maximum mining area.

However, if the mining area is extensive , the total mining area to be divided into sub-areas and they are to be opened up separately. Due to the wider and deeper reach draglines are the most suitable equipment for making a box cut. Standing on the surface they can load materials from the box cut and unload to the railway wagon.

RAMP

The number of ramp to be constructed must be as less as possible since it involves extraction of huge volume of rock mass and locks money in the form of developmental works. The stripping ratio and overall extraction are affected by the haul road geometry; and also it is very temporary in nature (since location of ramps change with time). Drop cuts

are used in each level to create a new bench. In case of softer rock (soil) ,dozer, shovel, backhoes, etc. can directly be used for notching the ground without the need of any further assistance but in case of moderately harder rock ripper may be used for ripping the ground prior to dozing/shoveling. However in case of hard ground drilling and blasting method must be applied. The notch is enlarged by taking successive cuts till the fully designed bench height is achieved. Once the full bench height is achieved, winning of the full faces with vertical/inclined blast holes proceeds in a sequence- upper benches to be advanced before the lower one in a chronological sequence. In the drop cut system dumpers may be placed either in one side or in both the sides. In case of loading from one side the shovel has to move along the two cutting paths (loading first from one bank, next shovel will move over and load from the other bank)(Fig 3.2).

But in the case when dumpers are placed on both the sides of the shovel, the latter is placed and moved through the centre line and dig from both the sides with a large swing angle to reach the dumpers. In this case the width of the ramp shall be more as compared to the former one for accommodating multiple dumpers (Fig. 3.3).

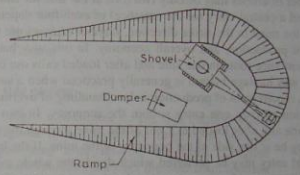


Fig. 3.2 : Ramp is being made with shovel and dumper combination (Loading by shovel on dumper is being done in one side only).

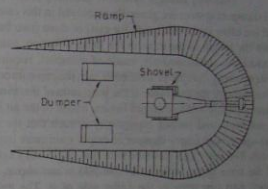


Fig. 3.3 : Ramp is being made with shovel and dumper combination (Loading by shovel on dumper is being done in both the sides)

Ramps may be constructed outside the planned pit boundary and in the waste. Under such a situation an additional quantity of rock equal to the volume of the ramp and road must be excavated. However if the ramp is placed inside the planned boundaries, a quantity of rock equal to the ramp and road volume will be left in place which may be extracted at the end of life of the pit. Some times the ramp along with the roads are constructed spirally through the contact point of country rock and ore or through the low grade ore body for economic reason. However the site specific conditions play the major role under this situation. After reaching the desired grade level or reaching to the optimum height of coal/lignite bench, the cut is expanded laterally. After creating sufficient space a number of mining equipment are placed on the bottom of the mine. The optimum face length are assigned to each equipment depending upon the size and type. After achieving the access route, the cut is widened up till the entyre bench/level is extended to the designed limit.

EXCAVATION OF OVERBURDEN

The removal of overburden is a very important operation in surface mining system. The method of excavation of overburden depends mainly upon the following main factor-thickness, dip and depth of the overburden rock mass, manner of occurrence of the deposits, i.e. whether the deposits are occurred under the flat surface terrain or over the hilly terrain, the surface topography, the ground condition, the environmental condition, production requirement, geotechnical parameters of rocks like the compressive strength, shear strength, tensile strength, modulus of elasticity, mineralogy, lamination, massiveness, toughness, geological disturbances, watery condition, the stability of the overburden and the spoil benches, etc. The selection of excavating equipment like dragline, shovel, front end loader, bucket wheel excavator, scraper along with ripper, surface continuous miner, etc, also depend upon the above geotechnical parameters of rocks. Physical characteristics like thickness, stratification, massiveness, scattered conditions, friability, hardness, looseness, watery condition, floor condition, the characteristics of geological structure, etc. are also very important in selection procedure. Climatic and other environmental conditions, rate of production requirement, expected life of the mine, combination of transporting means, the use of equipments either for production, overburden stripping or for both, the

distance of spoil disposal, etc. should be considered in great depth. The excavated overburden are disposed off either within the void created by the worked out area or away from the openpit. The number of entries (ramps, etc) to enter in and exit from the mine depend upon the following factors :

- Length of the mine in the strike direction,
- Amount of requirement of production,
- The requirement of separate routs for mineral and overburden transport,
- Distance of haul,
- System of handling of overburden soil, rock and also handling of mineral,
- Working of number of faces simultaneously,
- Overall economy,

One-way haulage system is generally preferred when a heavy amount of production is required. The truck and rail transport system are generally employed in this case. Number of entries may be only two (one at the side for entrance and the other in the other side for exit) or three (two for exit and a central entry for the entrance) or even four depending upon the length of the mine in the strike direction, number of the mine in the strike direction, number of faces are to be worked out simultaneously, mode of transport, requirement of production, and overall economy. In one way haulage system empty dumper or train enter into the mine through the entrance route in one side and after loaded exits out of the mine through the exit route at the other side of the mine. Two way haulage system is generally practiced when a surface mining pit is small to medium in size, length of the pit is small, requirement of production is low, handling of overburden is not high, and surface topography is such that the excavation of only one entry sustain the economy. In two way haulage system empty dumper or empty train enter into the mine and after loaded exit out of the mine through the same route. The entry into the mine with this single entry system may be at one side of the boundary of the mine. If the length of the mine in the strike is around 500 m and above, a central entry may be preferred which divide the whole mining block into two parts in the either side of it. The multiple pit entry and exit system involve heavy amount of earth excavation and hence it is very expensive. The location of the approach routs depend upon the following factors :

- Surface topography,
- Shape of the mine
- Length of the mine along the strike,
- System of waste disposal and disposing site,
- The required gradient of the haulage system,
- Depth of working,
- System of mineral and overburden handling,
- Requirement of production,
- Simultaneous working of number of faces in the mineral and overburden benches,
- Location of crushing plant,
- Overall economy,

Disposing off overburden within the extracted out void or around the periphery of the void are generally done by the following ways:

1. **Over-Casting or Side Casting by Shovel, Dragline and Bucket Wheel Excavators**

In this case the same excavator dig and load the spoil dump, swing towards the excavated void and fill it by unloading. Generally this system is practiced where the terrain is flat, deposit is flat, thin to medium thick and extensive in nature, the overlain overburden thickness is thin to medium thick, the excavation and production requirements are heavy, etc.

Over or side casting by dragline

The draglines are the common equipment for over-casting or side casting system because of their greater reach, greater flexibility, unaffected by the bank slides, water runoff and seepages and also if required boom can be extended for larger reach. The height and width of the bench should be the optimum to match the capacity of the dragline and its dumping radius. A stripping ratio around 2.5 : 1 (overburden: mineral) is very good for reclaiming the excavated area.

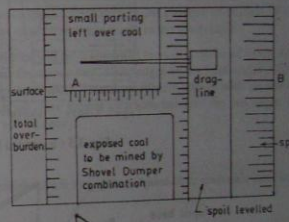
One dragline may be used for a single bench or for simultaneous operation of two benches. Sometimes combination of two draglines or one dragline and one shovel, etc. may be used for handling and rehandling of the spoil dump depending upon the condition. In this system working radius and the digging depth increase but at the cost of more capital expenditure on equipment and the cost of overburden over-casting. The multi-excavator system also facilitate to work a thicker bench height since one dragline will dig, load, swing and unload the overburden rocks at the reach of other dragline which will rehandling the same and dump the spoil into the void being reclaimed. A dragline can work in a bench voids of height more than 30m to 40 m.

If the working depth is more (100 m to 300 m) , handling of overburden is generally done with shovel and dumper combination or it may be done with the combination of shovel, mobile crusher, high angle belt conveyor and the belt conveyor system. The side casting system adopted by the draglines can be broadly grouped as following:

- Simple
- Extended bench
- Horse shoe
- Tandem

Simple side casting

Fig. 3.4 illustrates simple side casting system by a dragline. In this figure the dragline is installed over spoil heap after leveling the same. Dragline excavates overburden rock over the coal seam (portion A) and dump it over the spoil heap (portion B) for exposing the coal seam. The exposed coal will be extracted by separate benches with the shovel dumper combination.



Extended bench side casting

Fig. 3.5 illustrates extended bench side casting system by a dragline. The dragline is located over the leveled spoil heap and excavate the overburden rock over the coal seam and dump the same over the spoil dump for exposing coal seam. The exposed coal will be worked out separately with the shovel dumper combination. In this system dragline increases its reach at the cost of its productivity. While taking the soil/rock from the top; the efficiency of the dragline decreases by 70%.

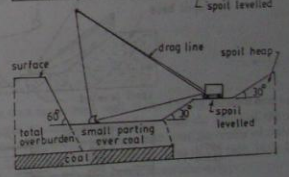


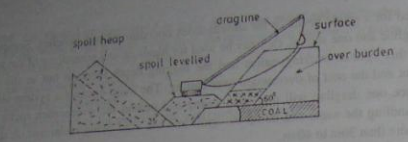
Fig. 3.4 : Simple side casting by the dragline

Tandem side casting

Fig. 3.6 illustrates a tandem side casting system by using two draglines. Drag.ine No. I is located over the surface which excavate overburden rock over the coal seam and dump it over spoil heap (Zone I). The dragline no. II is located over the leveled spoil which excavate the remaining overburden left over the coal seam and dump it on the spoil (Zone II) and also it rehandles the spoil of Zone I to dump it in proper place. The exposed coal will be worked out separately with the shovel dumper combination.

Tandem side casting

Fig. 3.6 illustrates a tandem side casting system by using two draglines. Dragline No. I is located over the surface which excavate overburden rock over the coal seam and dump it over spoil heap (Zone 1). The dragline no. II is located over the leveled spoil which excavate the remaining overburden left over the coal seam and dump it on the spoil (Zone II) and also it rehandles the spoil of Zone I to dump it in proper place. The exposed coal will be worked out separately with the shovel dumper combination.



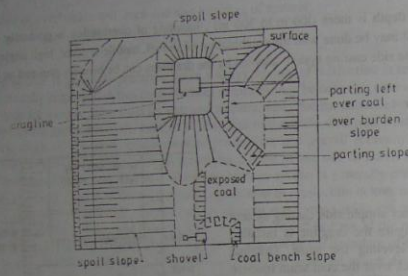
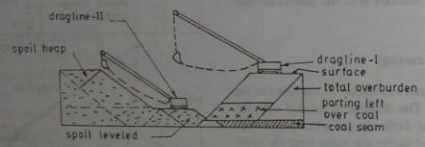


Fig. 3.5 : Extended bench side casting



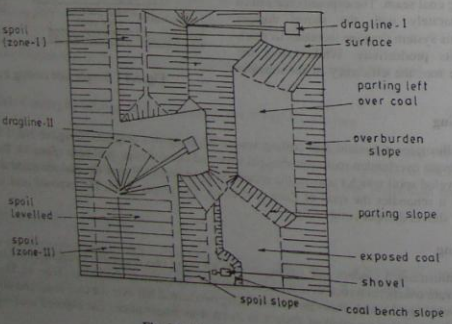


Fig. 3.6 : Tandem side casting

Over casting by shovel

It is not very common to overcast the overburden by a shovel since both its digging and dumping radius is short. However sometimes in a single overburden bench (thickness of the bench say around 15m to 20 m and after blasting, the blasted fragmented muck piles width around 35m to 40 m), blasted muck pile may be loaded and overcasted by a single or a double pass system after moving the shovel to a greater amount at the face to cover up the entyre face width. The overburden over the developed coal seams may be excavated by a hydraulic backhoe. Special precautions are to be taken in overcasting system by a shovel which are as following:

(1) The distance between the bottom edge of the mineral or coal and bottom edge of the spoil dump depends upon the mode of material transport system, the mode, manner and the size of fragmentation, production requirement, bench parameters, bench slope angle, the spoil dump slope angle, the watery condition and stickiness, presence of cementing materials and fines in the spoil, degree of required dilution factor, etc. In case one way traffic system, the distance between these two edges should be more to allow the trucks to pass through it.

(2) In case there is two way truck haulage system and the mineral is excavated directly by the shovel without drilling and blasting operation ,the clearance between the above mentioned edges may be zero since no traffic route is necessary in between them. But if there is no clearance space , there may be the chances of dilution of ore or coal with the overburden and there will be huge amount of mineral and coal losses in the spoil dump.

(3) In case mineral is to be blasted , the fragmented particles will be scattered at least in a radius of twice the width of the blasting face and hence if the clearance between the above mentioned edges is not kept wider , there will be higher chances of dilution of mineral with the overburden spoil. Therefore either a shovel with greater reach or a dragline will be preferred to overcast the spoil which will give greater clearance between the two edges. Otherwise not only the dilution but also mineral or coal losses will be heavy.

Cuts by Shovel

Basically two types of cuts are mainly counted through which the cuts are widened or materials are excavated by the shovels. They are as follows :-

i) Frontal cuts (Fig. 3.7 and 3.8)

ii) Parallel cuts (Fig.3.9 and 3.10)

Frontal cuts

In this system shovel faces the bench face, a niche cuts in the bank wall, starts digging forward and to the sides. Shovel loads materials over both the trucks/dumpers (one by one), positioned on both the sides of the shovel ,by swinging (average swing angle 60°). The shovel penetrates to a point in such a manner that the centre of swing shall be in line with the face. Shovel thereafter moves parallel to itself and takes another frontal cut in the face. With longer face having larger bench width, two to three shovels may be deployed in the same bench simultaneously.

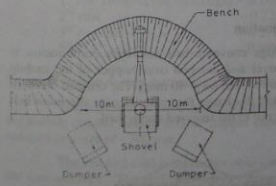


Fig. 3.7

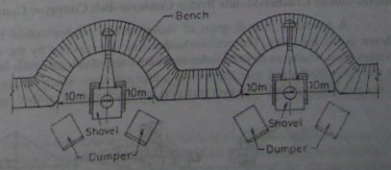


Fig. 3.8

Parallel cuts

This type of cuts are very common and are applied where there is the problems of space limitations particularly in the upper benches. In this case, there is only one access to the ramp from one side of the shovel and trucks approach to the shovel from the rear. Dumpers may be spotted only in one side or on both the sides depending upon the availability of space. The minimum width of operating room is equal to the width of the minimum required safety bench plus the width of the cut being winned. However the optimum width of working bench may be expressed in this way :

Distance from crest to berm + width of berm + 2m clearance + 0.5 x width of dumper + shovel centre line to truck centre line + maximum distance from shovel centre line.

In case of wider bench the slope angle will be flatter with the extra costs for stripping but the equipment operating efficiency will be higher. A narrow Working bench will offer a steeper overall slope at the cost of operating efficiency.

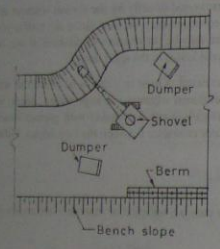


Fig. 3.9

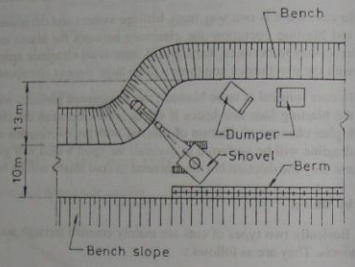


Fig. 3.10

Bench operation

In the Fig 1.1 it is shown that there are the presence of 4 benches. Operation starts from Bench-1. Drilling and blasting are to be done in the Bench-1. Shovel shall be moved and located over the floor of Bench-1. Shovel will load rock over the dumpers and dumpers will dump the same in the dump yard. The operation will continue till the optimum bench width is achieved. The cut, thereafter, to be taken from Bench-2 by drilling, blasting and loading by shovel (locating on the floor of bench-2). A portion of the Bench 2 will remain as a safety bench. The process shall be repeated until the bottom of the pit is reached. The shovel thereafter be moved back up to Bench-1 and the process shall be repeated in a cyclic manner.

Shovel-Mobile Crusher-Mobile Bridge Conveyor-Belt Conveyor Combination

A schematic line diagram of shovel-mobile crusher-mobile bridge conveyor-belt conveyor combination is shown in the Fig. 3.11. The ore/overburden rock is excavated by the shovel and dumped over hopper of the mobile crusher plant where ore or overburden rocks are crushed to a particular required size (say- 40 mm). The crushed ore/rock is thereafter discharged over the mobile bridge conveyor which in turn will discharge the material over the main belt conveyor. The main belt conveyor transport and discharge the crushed material to the required dump yard.

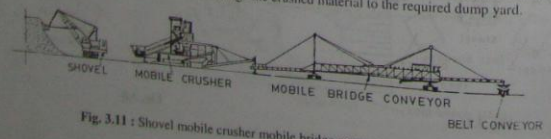


Fig. 3.11 : Shovel mobile crusher mobile bridge conveyor combination

EXAMPLE -1

Mobile In-pit Crushing Technology at Piparwar Opencast Coal Mines

The Piparwar opencast coal mines, situated in the North Karnapura Coalfield of India, first introduced the Mobile Inpit crushing technology in India. The Piparwar block is divided into 2 mining areas designated as quarry-1 and quarry-2 by a fault which has a - throw between 15m and 30 m towards the north west. The brief description about the project is as follows :

Mineable reserves - 197.49 Mt, Expected output (ROM) : 6.5 Mt/year

Stripping ratio (overburden in m^3 : Coal in t) = 0.65:1

Coal grade: Varies from E to G.

The name of the main coal seam under this mine is lower Dekra Coal seam whose thickness varies from 18.6m to 29.2 m. Average thickness : 22 m. The mining steps are as follows :-

i) Clearing of trees/bushes and removal of top soil : The combined thickness of the top soil and the alluvium is cleared off only immediately in advance of the mine to reduce the size of top soil dump , dust generation and also for ease of reclamation work.

ii) Removal of overburden: overburden rock is fractured by the drilling and blasting method .

250 mm diameter RBH is used to drill blastholes at a burden; spacing ratio 10 m: 10m. Bulk emulsion explosives are used taking with a powder factor of approximately 0.35 Kg/m^3. To avoid misfire double detonating cords are used in every hole. Time lag of detonating relays are 25 ms. Shovels with 10 m^3 bucket capacity, with a combination of 85 Tonne dumpers are deployed in the face. However thinner piles of inter- burden are excavated using 10 m^3 capacity Front End Loaders or a 9 m^3 bucket capacity Hydraulic shovel. Initially overburden rocks were stacked onto an out-of-pit-dump but presently the materials are being dumped directly in the decoaled area.

iii) Extraction of upper coal seams: After the drilling and blasting is over the same equipment used for the overburden are deployed for loading and transporting of coal. The coal thereafter unloaded to a hopper which thereafter will feed coal directly into the coal handling plant.

iv) Extraction of Lower Dakra coal seam: Mining of the Lower Dakra coal is carried out in strips of approximately 60 m to 80 m wide. A minimum of 60 m buffer is necessary between the coal face where the equipment (inpit crusher, belt wagon, belt conveyor, shovels, etc.) are placed for winning operation. Each strip is worked out in the up dip direction and as the shovel advances along the strip, the in-pit crusher and belt wagons are also relocate to follow the shovel. Each bench shall be at least 30 m wide. The bench floor gradient is 2^0.

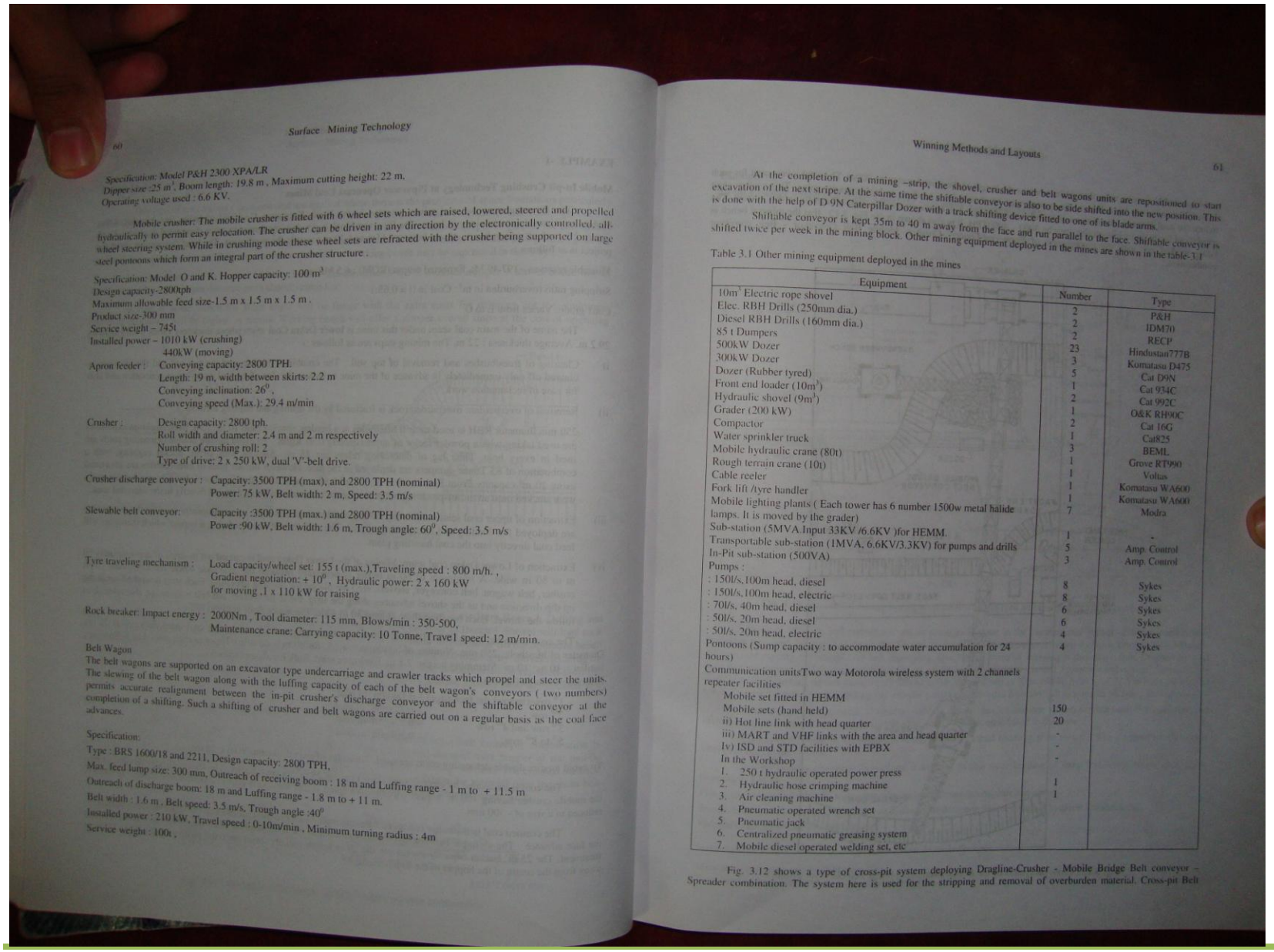
The coal is drilled by 1 DM- 701 R drill machine for a length of 23 m (average coal seam thickness is 22 m). Diameter of blasthole: 250 mm. Number of blasthole/row : 8,(for 80 m wide strip). Holes in each rows 4 to 5. Spacing : burden = 10 m : 10 m , Stemming height: 7-8 m, Explosive used: Bulk Emulsion explosive. Amount of charge (approx) :

1st row (nearest the face edge)	– 0.18kg/m^3 of coal
2nd row	– 0.22kg/m^3 of coal
3rd row and 4th row	– 0.27kg/m^3 of coal
5th to 8th row	– 0.30kg/m^3 of coal

To avoid misfire double detonating cords are used. Time lags of each detonating relays: 25 ms.

The coal is excavated by electric rope shovel having 25 m^3 bucket capacity. The shovel directly dump coal into the mobile crusher having 100 m^3 hopper capacity. An apron feeder transports coal to a double roll crusher where it is reduced to a size of - 300 mm.

The crushed coal is thereafter conveyed from the inpit crusher to the shiftable belt conveyor that runs parallel to the face advance . The shiftable belt conveyor transports coal to the coal handling plant for further size reduction and treatment. The 25 m^3 bucket capacity shovel can dig 25 m length radially from the fixed position and is kept 20-25 m away from the centre of the hopper of the in-pit-crusher.



Specification: Model P&H 2300 XPA/LR
Dipper size :25 m^3, Boom length: 19.8 m , Maximum cutting height: 22 m.
Operating voltage used : 6.6 KV.

Mobile crusher: The mobile crusher is fitted with 6 wheel sets which are raised, lowered, steered and propelled hydraulically to permit easy relocation. The crusher can be driven in any direction by the electronically controlled, all-wheel steering system. While in crushing mode these wheel sets are refracted with the crusher being supported on large steel pontoons which form an integral part of the crusher structure .

Specification: Model O and K. Hopper capacity: 100 m^3
Design capacity-2800tph
Maximum allowable feed size-1.5 m x 1.5 m x 1.5 m .
Product size-300 mm
Service weight – 745t
Installed power – 1010 kW (crushing)
440kW (moving)

Apron feeder : Conveying capacity: 2800 TPH.
Length: 19 m, width between skirts: 2.2 m
Conveying inclination: 26^0 ,
Conveying speed (Max.): 29.4 m/min

Crusher : Design capacity: 2800 tph.
Roll width and diameter: 2.4 m and 2 m respectively
Number of crushing roll: 2
Type of drive: 2 x 250 kW, dual 'V'-belt drive.

Crusher discharge conveyor : Capacity: 3500 TPH (max), and 2800 TPH (nominal)
Power: 75 kW, Belt width: 2 m, Speed: 3.5 m/s

Slewable belt conveyor: Capacity :3500 TPH (max.) and 2800 TPH (nominal)
Power :90 kW, Belt width: 1.6 m, Trough angle: 60^0, Speed: 3.5 m/s

Tyre traveling mechanism : Load capacity/wheel set: 155 t (max.),Traveling speed : 800 m/h. ,
Gradient negotiation: + 10^0 , Hydraulic power: 2 x 160 kW
for moving ,1 x 110 kW for raising

Rock breaker: Impact energy : 2000Nm , Tool diameter: 115 mm, Blows/min : 350-500,
Maintenance crane: Carrying capacity: 10 Tonne, Travel speed: 12 m/min.

Belt Wagon
The belt wagons are supported on an excavator type undercarriage and crawler tracks which propel and steer the units. The slewing of the belt wagon along with the luffing capacity of each of the belt wagon's conveyors (two numbers) permits accurate realignment between the in-pit crusher's discharge conveyor and the shiftable conveyor at the completion of a shifting. Such a shifting of crusher and belt wagons are carried out on a regular basis as the coal face advances.

Specification:
Type : BRS 1600/18 and 2211, Design capacity: 2800 TPH,
Max. feed lump size: 300 mm, Outreach of receiving boom : 18 m and Luffing range - 1 m to + 11.5 m
Outreach of discharge boom: 18 m and Luffing range - 1.8 m to + 11 m.
Belt width : 1.6 m , Belt speed: 3.5 m/s, Trough angle :40^0
Installed power : 210 kW, Travel speed : 0-10m/min , Minimum turning radius : 4m
Service weight : 100t ,

At the completion of a mining –strip, the shovel, crusher and belt wagons units are repositioned to start excavation of the next stripe. At the same time the shiftable conveyor is also to be side shifted into the new position. This is done with the help of D 9N Caterpillar Dozer with a track shifting device fitted to one of its blade arms.

Shiftable conveyor is kept 35m to 40 m away from the face and run parallel to the face. Shiftable conveyor is shifted twice per week in the mining block. Other mining equipment deployed in the mines are shown in the table- 3.1

Table 3.1 Other mining equipment deployed in the mines

Equipment	Number	Type
10m^3 Electric rope shovel	2	P&H
Elec. RBH Drills (250mm dia.)	2	IDM70
Diesel RBH Drills (160mm dia.)	2	RECP
85 t Dumpers	23	Hindustan777B
500kW Dozer	3	Komatasu D475
300kW Dozer	5	Cat D9N
Dozer (Rubber tyred)	1	Cat 934C
Front end loader (10m^3)	2	Cat 992C
Hydraulic shovel (9m^3)	1	O&K RH90C
Grader (200 kW)	2	Cat 16G
Compactor	1	Cat825
Water sprinkler truck	3	BEML
Mobile hydraulic crane (80t)	1	Grove RT990
Rough terrain crane (10t)	1	Voltas
Cable reeler	1	Komatasu WA600
Fork lift /tyre handler	1	Komatasu WA600
Mobile lighting plants (Each tower has 6 number 1500w metal halide lamps. It is moved by the grader)	7	Modra
Sub-station (5MVA Input 33KV /6.6KV)for HEMM.	1	-
Transportable sub-station (1MVA, 6.6KV/3.3KV) for pumps and drills	5	Amp. Control
In-Pit sub-station (500VA)	3	Amp. Control
Pumps :		
: 150l/s,100m head, diesel	8	Sykes
: 150l/s,100m head, electric	8	Sykes
: 70l/s, 40m head, diesel	6	Sykes
: 50l/s, 20m head, diesel	6	Sykes
: 50l/s, 20m head, electric	4	Sykes
Pontoons (Sump capacity : to accommodate water accumulation for 24 hours)	4	Sykes
Communication unitsTwo way Motorola wireless system with 2 channels repeater facilities		
Mobile set fitted in HEMM		
Mobile sets (hand held)	150	
ii) Hot line link with head quarter	20	
iii) MART and VHF links with the area and head quarter	-	
Iv) ISD and STD facilities with EPBX	-	
In the Workshop	-	
1. 250 t hydraulic operated power press		
2. Hydraulic hose crimping machine	1	
3. Air cleaning machine	1	
4. Pneumatic operated wrench set		
5. Pneumatic jack		
6. Centralized pneumatic greasing system		
7. Mobile diesel operated welding set, etc		

Fig. 3.12 shows a type of cross-pit system deploying Dragline-Crusher - Mobile Bridge Belt conveyor – Spreader combination. The system here is used for the stripping and removal of overburden material. Cross-pit Belt

conveyor length may be more than 610m whereas the length of the spreader may be 300m. The total investment for such an installation is very high. Hence, the system must be chosen for a medium to high capacity mine. This system shall reduce the haul distance, eliminates ramps and also high angle belt conveyor may be used efficiently in this system. The capacity of the in pit mobile crusher may be 3000 - 4000TPH. It is also shown in the Fig. 3.12 that the mineral bench is being worked out with the combination of Front-End-Loader -Crusher- Mobile Belt Conveyor - Belt conveyor combination.

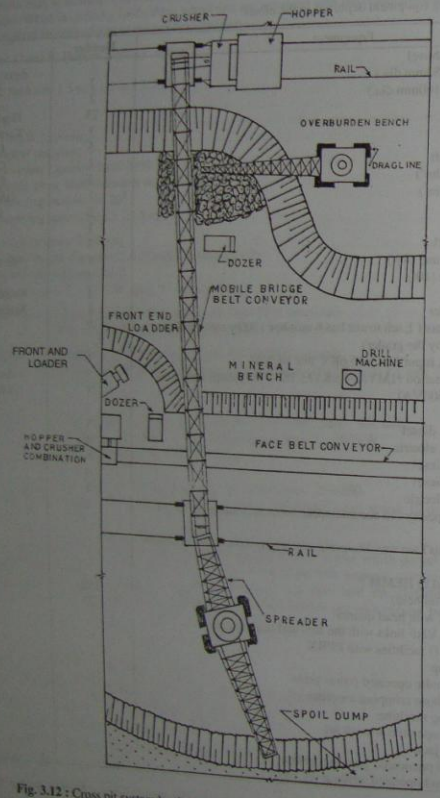


Fig. 3.12 : Cross pit system by the mobile overburden bridge belt conveyor system

Example : 2

In-pit Crusher –Steel Cord Belt Conveyor –Spreader Combination (Fig.3.13)

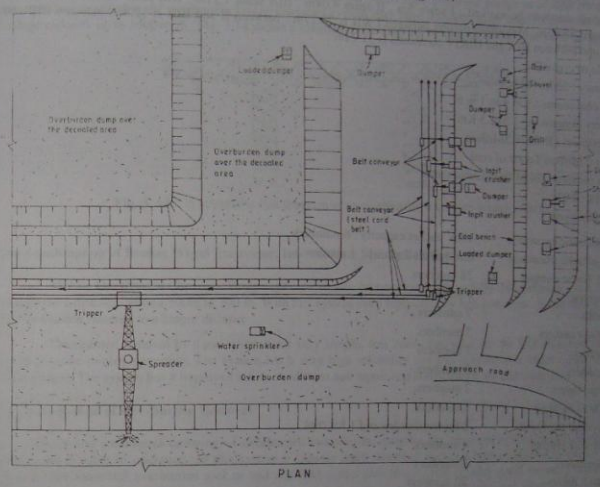


Fig. 3.13 : Layout of a mine having inpit crusher –steel cord belt conveyor-spreader combination

Drilling, blasting and excavation are as same as is done in the conventional Shovel –Dumper combination. Dumpers transport the material from the Shovel to the Crushers, where overburden / coal is crushed to - 300mm - 200mm size making it amenable for belt conveying. Overburden material crushing is carried out from Crushers by steel cord belt conveyors and then dumped out inside pit through the Tripper car and Spreader combination. Spreaders are crawler track mounted and can dump to a width of 100m, down dump of 30 m and high dump of 22.5 m from the crawler level. Tripper car provides for shifting the transfer point of overburden from the belt to spreader all along the conveyor.

There are 3 sub - systems, 2 systems serving the overburden and one system serving coal. For operation and maintenance convenience, a conveyor distribution point (COP) is established in the mine.

Coal after crushing in the project is being carried to a coal handling plant by a 6.3 Km length of overland coal steel cord conveying system.

In - pit crusher conveyor technology is selected in this mine because of steep inclination or the coal seams (dipping at 1 in 4).

This is environmental friendly technology due to less exhaust fumes, dust and noise.

This technology has got much reduced lead distance for dumpers, hence reduction in diesel consumption.

Volume of accidents are also very less.

Some Salient Points

Mine able coal reserves : 39.28Million Tonne

Total overburden to be removed : 196 Million bulk m^3

Planned area of excavation : 226 Hectares
Life of mine : 25 years
Number of coal seams to be worked : I seam 5m , II seam 4.04m , IIIA seam 1.21m , IIIB seam 1.37m and III &IV combined seam 15.63m. Maximum parting between I &II is 23.17m , between II and IIIB is 12.8m, between IIIB and IIIA is 31m and between IIIA and III&IV combined is 31.71m .
Overall grade of coal : D
Depth of quarry : 220m (Max.)
Gradient of coal seams : 1 in 4 to 1 in 6
Present quarry dimension : 1.5x1.3 Km
Capital investment : Rs8078.5Million
Production : 2 Million Tonne of coal
Overburden removal : 70 Million Cubic meter
Overall OMS : 4.97
Equipment deployed

1.	Hydraulic excavator	: $10m^3$ bucket capacity : 6 numbers
2.	Hydraulic excavator	: $8m^3$ bucket capacity : 1 number
3.	85 Tonne dumper	: 28 numbers
4.	Dozers:	: 7 numbers
5.	Drills	: 7 numbers
6.	Water tankers	: 3 numbers
7.	Graders	: 3 numbers
8.	Loaders	: 2 numbers
9.	Pipe layers	: 2 numbers
10.	Cleaning plate	: 1 numbers
11.	System conveyor	: 2 for overburden and 1 for coal
12.	Mobile crusher	: 4 numbers
13.	Spreaders	: 3 numbers
14.	Belt drives	: 20 numbers

Overcasting by Bucket Wheel Excavator (BWE)

The bucket wheel excavator along with the combination of spreader can effectively be used for overcasting the loose, soft and thick overburden. However hard rocks should be pre-blasted with the help of light charge for fragmentation. A BWE gives continuous output, it has low operating and power cost, excellent loading as well as dumping (directly to the spoil dump) characteristics and facilities. A combination of shovel for removing and dumping softer layers of the strata over the harder rock boulders (discharged by the shovel) may also be employed if necessary. The continuous bucket wheel excavator system may have the following combinations-

- Bucket Wheel Excavator - Conveyor Bridge
- Bucket Wheel Excavator- Intermediate Mobile Belt Conveyor
- Cross pit system (Bucket Wheel Excavator- Intermediate Mobile Belt Conveyor- Stacker)
- Bucket Wheel Excavator- Double Line Locomotive System

Generally the rail mounted BWE is predominantly used compare to a crawler track mounted BWE.
There are various possibilities of methods of working with the help of a BWE which are as following :

Overburden Removal by the Bucket Chain Excavator or BWE and Overburden Conveyor Bridge Combination

With the Bucket chain excavator and overburden conveyor bridge combination, overburden can be transported directly and discharged around a distance of 700 m away from the face. The excavating thickness with this system depends on the span and the length of the discharge boom. The. F-60 overburden conveyor bridge system has a service

weight around 15900 t and capacity 120 million m^3. Each overburden conveyor bridge is connected with 3 bucket chain excavators of total weight around 14000 t. Therefore the total system weight is around 30000 t. The system can steadily move at the bench in the overburden excavation. Height of the overburden conveyor bridge is around 60 m. The belt conveyor incorporated in the overburden bridge conveyor system may have a speed more than 10 m/s and a theoretical haulage capacity up to 3600 m^3/h. Fig. 3.14 shows a schematic line diagram of an overburden conveyor bridge system.

Fig. 3. 14 : A schematic line diagram of an overburden conveyor bridge system.

The Combination of Bucket Wheel Excavator, Intermediate Mobile Belt Conveyor

Mobile Overburden Conveyor Bridge system can also be employed successfully for continuous soft overburden removal to a distance around 500m to 700 m. With this system even high capacity belt conveyor can be linked to dispose off overburden material to a longer distance.

Overburden removal by these systems are very simple, less costly and used for the less disturbed flat deposits for large scale mining operation. The system has a very high reliability and can operate at a wider range of operating environment. The system has a improved safety conditions and repair and maintenance are also very easy.

Cross Pit System of Overburden Removal

Direct and continuous dumping of overburden can also be done by the cross pit system which is very simple, cheap, and flexible in operation. In this system a crawler track mounted bucket wheel excavator is employed which excavate the loose/soft overburden rock or soil and feed onto a slewable mobile crawler track mounted stacker system. The stacker there after discharges the overburden over an extra-long discharge boom and finally dump it over the overburden dumping place (Fig. 3.15). The bucket wheel excavator stays over the overburden of the exposed lignite seams/other soft mineral deposits like mineral salt, etc. The cross pit system expose large amount of mineral deposit at a larger span. The lignite will be excavated below the stacker system, loaded and hauled by means of a separate conveying system. The system can be efficiently applied in less geologically disturbed and flat deposits.

The cross pit system can employ bucket wheel excavator of bucket volume ranging from 65 m^3 to 4500 m^3 and having power and capacity ranging from 45kW to 2680 kW and 300 m^3/h to 18000 m^3/h respectively. The stacker which can be incorporated with the system may have boom length and hourly transporting capacity varying from 30m to 225 m and 500 m^3/h to 23000 m^3/h respectively. A material handling of 40 million cubic meter of overburden per annum is not uncommon nowadays. An intermediate mobile transfer conveyor may be placed in between the bucket wheel excavator and the stacker system when the working levels (elevations) of BWE and the stackers are different. This facilitate the extraction of mineral of higher thickness and better and safe negotiation of rough ground profile. The excavation thickness with the cross pit system is around 40 m. Table 3.2 and 3.3 will give brief specifications of bucket wheel excavators and stackers respectively whereas Table 3.4 will show some geotechnical data for bucket wheel excavator environment.

Overburden from the initial mine cut is generally dumped outside the mining area till enough void is created inside the mine by the extraction of mineral. For overcasting spoil ,bucket wheel excavator (BWE) in conjunction with belt conveyor and spreader may be employed. Overburden (very soft and loose) may be removed with the help of BWE (bucket capacity 1.4 m^3) belt conveyor (2000 mm belt width) and spreader (11000 TPH) system. In place of spreader mobile transfer conveyor may be employed. Bench to bench link is possible through dumping conveyor system by means of telescopic drive head. Development may be done on swinging pattern to swept the whole mining block in one swinging operation. The overburden is taken out through the swinging point. The overburden benches and the mineral

Bench conveyors may have interlinking arrangements as a common platform termed as inter-linking platform and the interlinking between the system is generally done through telescopic drive heads. This is done generally for greater flexibility and inter-changeability of operation in the overburden benches. Initially spoils from all the benches are brought to the outside spoil bank segments. During back filling the spoil into the excavated area, first the overburden rock is refilled and thereafter soil is spread over it for total reclamation.

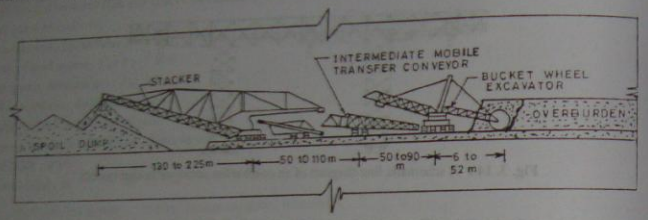


Fig. 3. 15 : A generalized bucket wheel excavator and stacker combination for overburden removal .

Working methods by bucket wheel excavators

The following method of workings can be practiced by a bucket wheel Excavator (BWE):

- Face or front working
- Full block working
- Half block working

All the above mentioned workings can be done by (i) Terrace cut (Fig. 6.10) (ii) Dropping cut (Fig. 3.16) or by (iii) The combination of both.

Face or front working

In this case rail mounted bucket wheel excavator digs the high cut either by terracing or by dropping cuts or by a combination of both and BWE travels along the face. With the BWE layers of soil, sand strata, lignite, etc. can be excavated efficiently.

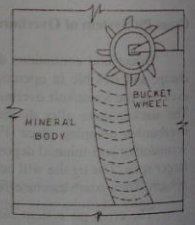


Fig. 3. 16 : Dropping cut

Full block working

A block of face can be excavated in several benches cuts by raising, lowering and slewing continuously the wheel of the long boomed BWE which may have long intermediate bridge. From face, the excavated material may be transported by the shiftable face belt conveyors with or without crawler-mounted belt structure or by directly dumping it onto the surface dump yard. The excavator with boom crowd sets the depth of each slice cut by pushing forward the boom of the bucket wheel along a track in truss of the heavy structure. The crowdless BWE cuts the slice depth by advancing the crawlers .Crowd type BWE can cut horizontally up to an angle of 90° w.r.t. the axis of the excavator path whereas the crowdless excavator has to reduce its slewing angle in the lower cut . However, both the crowdless or crowd type machine can cut a bench slope down to 38° (Fig. 3.17 and 3.18).

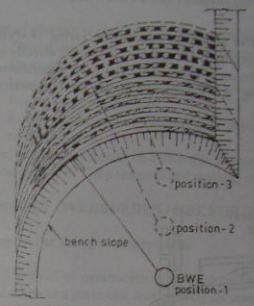


Fig. 3. 17 : Stickle shape cut full block working by a bucker wheel excavator (crowdless type)

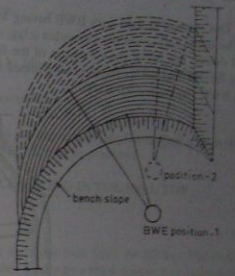


Fig. 3.18 : Full block working by a bucker wheel excavator (crowd type)

Table 3.2 shows the specification of a few bucket wheel excavators manufactured by TAKRAF, Germany, whereas the Table 3.3 shows the specifications of a few Stackers manufactured by the same company i.e. TAKRAF, Germany.

Table 3.2 Specification of a few bucket wheel excavators (manufactured by TAKRAF, Germany)

Description Type	SRs. 65	S Rs 240	S Rs 400	S Rs (H) 702	S Rs (K) 2000	S Rs 4000	S Rs 6300	S Rs 8000
Drive Power (kW)	45	160	500	400	2 x 630	3 x 700	3 x 630	4 x 670
Theoretical Haulage Capacity (m^3/h)	300	1150/850	2800/2300	2500	4500/3600	11000	14000/10500	18000/9900
Excavating height/depth (m)	6.0/3.0	9.5/0.5	14.0/1.0	20/2.5	28/3.0	36/3.5	50/15.0	50/7.9
Specific digging force (N/cm)	436	470/650	910/1125	680/150	2640/3340	1440	710/1130	970/1670
Service weight (t)	47	142	500	920	2350	5100	8950	9500
Bucket volume (m^3)	65	240	550	550	350	3200	3300	4500
Discharge rate, m^3 per minute	78	78/58	84/70	95/19	216/172	57.5	71/53	67/37
Caterpillar ground Pressure (N/cm^2)	7.0	9.0	11.0	11.0	11.0	12.6	14.8	13.0

Table 3.3 Specifications of a few stackers (manufactured by TAKRAF, Germany)

Type	Theoretical haulage	Length of conveyor	Dumping height (m)	Service weight (t)
A Rs 550.30	550	30	9.5	95
A2 Rs 2500.105	2500	105	25	1150
A Rs (k) 5500.95	5500	95	33	1746
A Rs (k) 8800.195	8800	195	60	3700
A2 Rs 12500.225	12500	225	30	6300
A2 Rs 18000.120	18000	120	33	2990
A2 Rs 23000.130	23000	130	33	3665

Half block working

This method is used with BWE having long boom for selective mining of minerals occurring in several thin seams. The upper soft layers of overburden (clay, sand, etc.) are removed by BWE and are overcasted directly onto the top spoil dump. The BWE travels on top of the lignite body with a boom crowd action. The hard minerals/rocks are generally blasted and are excavated and dumped by large shovels (Fig. 3.19). Table 3.4 shows the parameters of the compact BWE and the normal BWE.

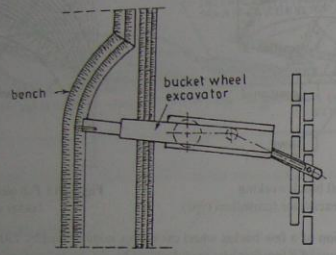


Fig. 3. 19 : Excavation using face block or side block technique

Table 3. 4 Geo-technical environment of 2 types of bucket wheel excavator

Sl. No.	Description	Compact BWE	Normal BWE
1.	Workable thickness, (m)	15	24
2.	Breadth of a block, (m)	20	45
3.	Relative partial block volume	1	10
4.	Theoretical haulage capacity, (m^3 /h)	3150	3150
5.	Effective haulage capacity (m^3 /h)	1420	1770
6.	Efficiency of BWE (5 / 4)	0.45	0.56
7.	Service weight (t)	480	1150

Piling up of overburden around the periphery of the opencast mine

This arrangement is generally practiced under the following conditions-

- Non-availability of the dragline or shovel or BWE with longer reach to overcast the overburden spoil.
- Non-availability of the proper dumping area away from the mine,
- Amount of overburden to be handled is heavy,
- Thickness of the mineral body is very high which requires a number of benches to be developed to extract the complete thickness of the mineral body,
- In steep mineral body,
- Economy of the transport distance.

Generally in this case shovel, dragline, front end loader, scraper (for thin and softer rock formation), etc are used as excavating machinery whereas dumper, rail and scraper (for smaller haul distance) are used as transporting media.

Disposal of overburden rock away from the pit

This type of practice is generally adopted under the following conditions:

- The deposits are thick and steeply inclined,
- Deposits occur in the hilly terrain,
- Ponds, lakes, any unproductive low lying topography area to be filled up for special use or cultivation,
- For making embankment,
- Extensive removal of overburden is desired to obtain huge amount of production,
- In the deep pits.

However, this system should be avoided in the comparatively thick and flat deposits having flat surface terrain since it is a very costly system. Only removal and stacking of top soil away from the pit may be practiced.

DISPOSAL OF OVERBURDEN / ORE FOR THE DEPOSITS OCCURRING IN THE HILL TOP

By filling up the valley or depressions

Disposal of overburden is easily done by simply discharging the waste rock from the hill top. The overburden rocks are taken to a particular side running all along the face length either simply by a truck road or by a railway line and are dumped to fill up the valley or depressions for reclamation. The disposal of overburden directly in the downhill side will cause degradation of mining environment like soil erosion, water pollution, ecological imbalance, destruction of trees and fertile hill slope land, etc. The pushing of the spoils in the downhill side should be in the steeper portion and to be done in a controlled manner to fill up the valleys near to the mine away from the cascade, seepage or drainage of water from other sources. Before filling the valley the quantity of swelled overburden volume is to be determined to fill up the valley as well as disposing it off outside the extracted area. The valley is to be prepared first to accept the dump material. Then top soil from the hill top area is to be stripped out and transported and stacked in a separate place away from the valley. After making a trench in a suitable place the overburden material is dumped over that prepared area. When it is filled to a particular height, it will be graded by a bulldozer and the stacked top soil is again to be sprayed over the graded land for proper re-vegetation. Slope of the reclaimed area in all direction should be within 25° from the horizontal.

The reclaimed land should also be provided with ditches at close intervals to divert surface run off water in a planned and controlled manner and also to protect the soil erosion. After reclamation is over, it will immediately be vegetated with trees and herbs for flower, fruits, wood, etc.

By vertical or inclined shafts and drifts

In steep hilly terrain there are severe restrictions on environmental pollution like vibration and noise due to plying of heavy trucks and vehicles, production of noise by running of several mining equipment, air, water, and land pollutions, etc. However where a heavy amount of production is necessary ,a vertical or inclined shaft around 6m to 10 m in diameter (around 5 times the size of largest piece of blasted rock/ mineral) may be sunk through the ore body at an optimum location. The shaft ends at the foothill of the mountain where a level tunnel drifted from the foot of the mountain may be connected to the bottom mouth of the shaft fitted

with a chute. A primary crusher is installed near the top mouth of the shaft to crush the rock or ore (dumped into it by dumpers) to a particular size and discharge them into the shaft. The crushed ore or rock is thereafter transported either by a belt conveyor or rail installed in the tunnel or sometimes even they may be transported by the dumper to the primary stockpile. The ore or rock thereafter re-crushed in the secondary crusher installed near to the primary stockpile and be transported to the required destination. The size of the ore pass depends upon the geotechnical parameters of rock and the ore, the height of fall, storage capacity, rate of inflow and discharge, the climatic conditions, etc. Both the sinking of shaft or incline and drift (tunnel) are very expensive operation, hence before considering this methodology a full techno-economic analysis (type, size, shape, physical characteristics, dip of the ore body, competency of rock, surface terrain, free flowing characteristics of ore, other geotechnical parameters of the rock and ore, compatibility and profitability of the system as compared to the other system, etc.) to be carried out. A production of around 3000 - 15000 Tonne of ore per day is easily achievable with this system.

TRANSPORTING OF OVERBURDEN / ORE IN STEEPLY DIPPING DEPOSITS

In case of steeply dipping massive deposits occurred at great depth in a relatively smaller lateral extend where conventional type layouts are not possible or where making of approach road involves heavy cost, the following transport media may be suitable-

(i) Mobile crusher - high angle belt conveyor-belt conveyor system,
(ii) Skip,
(iii) Vertical or inclined shaft connecting each of the benches with one drift for the free flowing of ore or rock.

Mobile Crusher-High Angle Belt Conveyor-Belt Conveyor System (Fig. 3.20)

In this system the blasted ore or rock is fed in a mobile or semi mobile crusher located in the surface mining system where the ore or rock is primarily crushed to the required size. The crushed materials are discharged onto a high angle belt conveyor which discharges material in turn onto a long belt conveyor . Fig. 3.21 shows a retractable boom type high angle conveyor.

The belt conveyor discharges the material onto a secondary crusher plant and finally the ore or rock are conveyed to the required dump yard.

The mobile crusher generally contains gyratory crusher but in some of the cases even jaw crusher are employed. This type of mobile crushers is generally crawler track mounted or rail mounted or has walking mechanism. They also incorporate electric motor, feed mechanism, overhead crane, discharge hopper with a feeder, lubrication system,, various control units, dust control mechanism, etc.

Although the system is less flexible and incapable to carry larger size ore or rocks but it provides the following advantages-

- As the depth of working increases this system offers cheaper means of transport system as compared to a truck haulage system,
- Manpower requirement, operating and maintenance cost, fuel cost, etc, are very less in this system as compared to a truck haulage system. Consumption of fuel in the truck haulage system will be tremendous high in a very deep mine.
- This system requires less excavation for transport route. Hence construction and maintenance cost is very less in this case.
- Initial investment in this system is more but ultimately it is more economic since it's operating and maintenance costs are cheaper as compared to a truck haulage system,
- This system has longer life, high production capacity, and high energy utilization efficiency,
- A high angle conveyor can negotiate an angle of more than 60° whereas an ordinary belt conveyor can negotiate a gradient of around 18° to transport materials,
- Production of noise, vibration and dust will be less,
- Control of dust is very easy and they are not very much affected by the climatic conditions as compared to a truck haulage system.

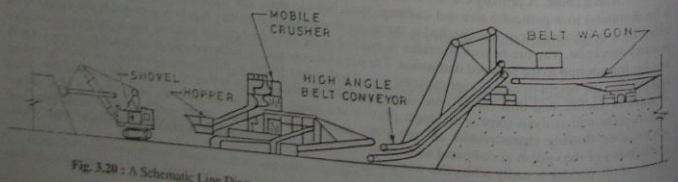


Fig. 3.20 : A Schematic Line Diagram of Shovel, Mobile Crusher, High Angle Belt Conveyor.

Belt Wagon and Belt Conveyor Combination.

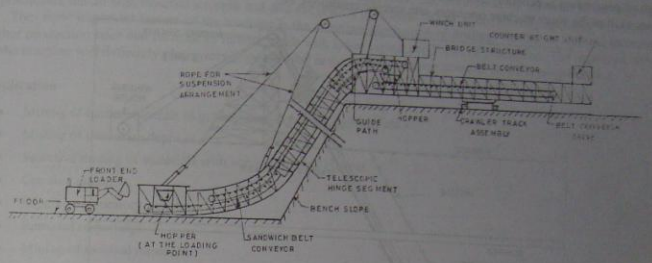


Fig. 3.21: Retractable boom type high angle conveyor.

Skip Transport (Fig. 3.22)

It is a transporting system in which a properly designed skip meant for ore or mineral and of sufficient size is raised up or lowered down by the hoist system either through the shaft or incline. The skip is either bottom dumped or side dumped. The loading onto the skip is done from the skip pocket or bins either from one or more points and after hoisting they are dumped onto the bins or plate conveyor-belt conveyor -crusher- belt conveyor system placed at the top of the surface or in the intermediate positions. The skips are hoisted either by single drum or double drum winding engines. Inclined skip hoisting system is practiced in the surface mining operation for small steeply dipping, or massive deposits occurring at a great depth, where very long steeply dipping permanent truck routs are very difficult to maintain and also where flexibility is less.

The skips are used to reduce the stripping ratio to eliminate the haulage road and it's maintenance, and also it increases the production and productivity. Since number of haulage equipment are less in this case their operating and maintenance costs are also less. The capacity of this type skip hoisting system depends upon the production capacity, mining and haulage layout, size and design of skip, size, stickiness and nature of the ore, capital cost of the equipment, installation, maintenance and operation, hoisting speed, cycle time for the complete operation, surge capacity, possibilities in future extension, etc. It is again worth to mention that in the surface mining standard rail and tie mounted skip of capacity ranging from 10 Tonne to more than 35 Tonne with a power rating from 1000 kW to 1600 kW and capacity varying from 500 Tonne to 1500 Tonne per hour are not uncommon. During material handling system it must be properly noted that the feed size should never be oversized, material should not be sticky and abrasive. Oversize ore should be crushed on the surface to a particular size varying from 1 m to 1.5 m and a grizzly should be used to screen out the oversized ore or rock. In case of abrasive materials a special manganese steel liners may be lined into the skip. The loading onto the skip may be done by permanent (installing, moving, scrolling to control the flow of materials from various level), semi-permanent (this structural fabrication is lowered down successively as the mining activity progress) and mobile arrangement to load onto the skip from various levels. All the controls for acceleration, retardation, run, dumping arrangement, opening and closing the skip door, etc. are automatic. Presently failsafe protections are incorporated to prevent overload, over speed, derailment, control of slack rope, etc.

AUGER MINING OPERATION IN THE SURFACE COAL MINES

When the faces reach to the points of maximum stripping ratio and when further advancing of faces are stopped, the auger mining can be practiced as a secondary extraction of coal and other softer mineral bodies. Generally when opencast mining operation reaches to the boundary of the mine and the benches are stopped in such a manner that an ultimate pit slope around 40° to 45° is obtained, this auger mining is done since with this system no overburden

excavation is necessary. Auger is driven into the coal/other softer mineral bodies which are exposed by the high-wall. Depth of penetration is generally 50m to 60 m depending upon the geotechnical parameters, thickness of seam/ mineral bodies, skill of the auger machine operators, site specific conditions, etc.

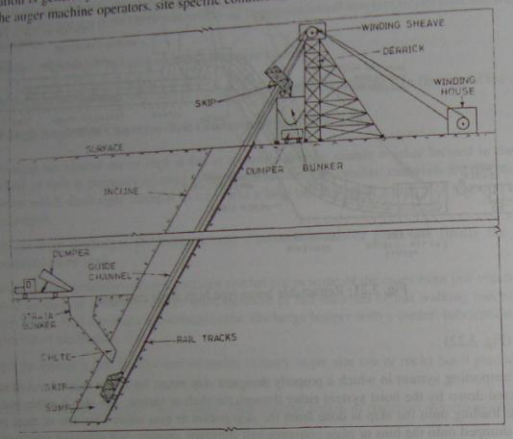


Fig. 3. 22 : A schematic layout of a skip.

An auger in the surface mining operation is a skid mounted machine in which there may be the mounting of a single auger head of diameter 0.45 m to 1.20 m ,two auger-heads of diameter 0.45 m to 1.0 m each, or three auger heads of diameter 0.45 m to 0.6 m each. The special type of cutting head is attached with the scroll rod having length varying from 2 m to more than 10 m. The machine with the help of electric motor coupling and gear box system rotates the drill scroll rod to penetrate into the coal seam and the cut coal comes out from the drilled hole. The cut coal by the auger is carried out by the auger flights and transferred to the conveyor belt which is towed to the machine. The machine conveyor belt thereafter transfer coal on to a belt conveyor or over a dumper. If dumpers are employed in the mine , coal or ore may be dumped over the floor by the auger and afterward it shall be loaded on to the dumpers by front-end-loader. During making holes by the auger care must be taken on sufficient partings to be left in between the two holes otherwise there may be the chances of caving of overburden. At the same time it must also be taken into consideration that the productivity and recovery ratio are very high (in the order of 50% to 60%).

The scroll type drill rod can drill a length of 50 m to 60m into the coal seam. After drilling one hole, another hole can be made by it's side leaving a rib around 150 - 200 mm in between them. Output per hour with this type of auger machine is not very much. It is around 500 Tonne per hour.

BLAST HOLE DRILLS

Table 3.5 shows the brief specification of some of the blast hole drills for modern opencast mines.

SURFACE CONTINUOUS MINER IN OPEN CAST MINING TECHNOLOGY

From soft rock continuous mining by the bucket wheel excavators, the current state-of-art of opencast mining technology is continuously shifting towards mining of harder materials like coal, gypsum, limestone, etc. by the recently developed continuous surface miners. These highly powered continuous surface miners not only offer continuity of

operation in hard rock mining but also can be used for selective mining of thin seams, for loosening of consolidated layers being a supporting equipment for the bucket wheel excavator; for excavation of mineral where ripping is impracticable due to high material strength and also excavation of mineral blocks where blasting operation is prohibited; etc. The most important feature of the machine is the complete elimination of drilling, blasting or ripping; increase of higher production rates and increase of mining yields with lower capital per Tonne of production. The improved version of the machine will definitely play a very important roll in the opencast mining scenario in the coming decades.

Application

- Mining of harder minerals (e.g. limestone, dolomite, gypsum, etc.)
- Mining of thin seam deposits.
- Selective mining of materials with varying mineral content.
- Creating channels.
- Digging exploratory trenches.
- Removal of partings.
- Mining of residual minerals.
- Removal of consolidated overburden layers.
- Road construction and maintenance.
- Digging drainage ditches.
- Moving conveyor units with an attached side boom

Types

The following three types of surface continuous miners are available in the world market :

- Bucket wheel type : Cutter picks are fitted on various bucket on the cutting drum, which is situated in the front of the machine. The machine can exert huge force in cutting both hard coal and rock. This type of surface miner is basically used for mass production.
- Milling type : Cutter picks are fitted on drums and depth of cut is up to 600 mm (max). Production capacity is low and can be used mainly for selective mining.
- Boom type : Cutting drum fitted with picks is mounted on a boom, like road headers. Due to longer boom the cutting force of the machine is limited.

Mechanism of Rock Excavation by Bucket Wheel Type Surface Continuous Miner

The mechanism of primary rock excavation with this type of continuous surface miner is basically penetration of a wedge of some shape into the material to be cut and this action is performed by a heavy duty drum equipped with large number of tungsten carbide tipped cutting tools. The process of digging is commonly specified by the resistance to digging:

$D = (P/w) \times t$ (kgf/cm^2),

where,

D = Digging resistance, kgf/cm^2

P = Tangential resistance force of rock for digging, kgf.

w = Width of excavated block, cm

t = Depth of cut, cm

Table : 3.5 Some blasthole drills for modern opencast mines

Model	Make	Collaboration	Hole size (mm)
SBSH-250	Heavy Engg. Corporation, India	USSR	243-269
IDM-70E	Ingersol-Rand (India) Ltd., India	I. Rand	235-265
ICM-10	-do-	-do-	102-114
IDM-30	-do-	-do-	160-200
C 750 E	Revati C. P. Equipment Ltd., India	CPT	229-269
C-650	-do-	-do-	160-200
RECP-750	-do-	-do-	250-310
SK-60	LMP Precision Engg. Co. (P) Ltd., India	Reedrill	200-280
RBH-1000D	-do-	-do-	160-200
CD-40	Krishna Rock Drill Pvt. Ltd., India	-	102-114
WDT-350	Water Development Society, India	-	105-165

It can be noted that Tamrock of Finland is manufacturing percussive drills of type 4000 which canattach drill bits in the range between 178mm to 254 mm, whereas Gardner-Denver of USA is making rotarydrill which can accommodate drill bit of size 311mm to 559 mm.

D and P depend on the compressive , shear and tensile strength of rock, its density and also on jointing property of the rock mass that is characterized by the co-efficient of structural weakening of rocks in the direction of digging. The value of structural weakening depend upon average size of elementary joints and number of joints. D is reduced by multi-bucket excavators fitted with sharp cutting edge and also increasing the number of simultaneously cutting elements and decreasing the spacing of cutting elements. D is also reduced for semi-free/free digging than blocked digging. Depth of cut is determined from the diameter of the cutting drum. Overall cross-section of the cut depends upon size of cutting drum, power of the machine, weight of the machine and also on the configuration, design and material of cutting elements.

KSM2000 surface continuous miner manufactured by Krupp Fordertechnik , Germany ,comes under this category . It is a crawler mounted continuously operating mobile opencast mining machine, which cuts consolidated soils. The KSM 2000 attains its nominal capacity in rock up to a compressive strength of 40 MPa. While cutting, the material is pre-crushed to the size (-100 mm) which is very much suitable for belt conveying, The excavated material is transferred to downstream means of transportation either by dumpers or by conveyors. The specialty of KSM 2000 is the provision of parallel bucket wheels in front of the machine that is attached to main frame without a boom. The cutters of bucket wheels are equipped with point attack picks, which cut the mining face during the continuous advance of two-crawler track assembly. A dozer blade is arranged behind the bucket wheels which are constantly pressed to the ground during cutting operation. The machine is built for high capacity digging up to 2.9 m cutting depth and a block width of 7.1 m. KSM 2000 is primarily a production machine which can be compared with the present conventional mining system. The design of KSM 2000 also permits the selective mining of changing coal/rock layers of multi-layer deposits with small percentage of fines. In case of selective mining , the machine is able to maintain its nominal capacity up to a depth of cut of 800 mm by suitably increasing its advancing speed. The output of KSM 2000 amounts to 1400 m^3/h in continuous cutting operation under normal conditions for rockup to a compressive strength of 40 MPa . By turning the bucket wheels in front of the machine and continuously advancing the crawlers, the face in front of the bucket wheels is cut by the picks. The buckets pick up the cut material directly from the mining face and discharge it at the rear side of the bucket wheels by gravity on to chain conveyors. The conveying system of KSM 2000 consists of two short lead-in chain conveyors, the main belt conveyor and the belt conveyor of discharge boom. The discharge conveyor is provided with slewing (±90°), luffing (high 12.6 m at +25° and low 3.9 m at -8°) arrangements for facilitating discharge either onto dumpers or on conveyor systems.

The benches are cut in layers from top to bottom and each layer in strips. Depending on the initial state in the opencast, each layer is extracted by terrace cut or cut by the KSM 2000 initially cuts a channel and subsequently removes the rest of the layer with terrace cuts. The width of cut is 7.1m. The depth of cut may be varied from 0.5m to 2.9m. The normal cut dimension is 7.1m by 20.4m. Cut height and advance speed of KSM 2000 can be continuously adjusted. Travel speed during cutting can be adjusted between 0.37 - 3.0m/min. Relocating speed in level terrain may be

varied between 1.36 - 31.9m/min. In normal operation KSM 2000 advances at 1.36m/min. The speed of bucket wheel is fixed at 6 rpm.

This machine can be used for the following :

1. Primary mining
2. Selective mining of layers exceeding 500mm
3. Selective mining with high separation accuracy
4. For production of less fines but good lumps (-100mm) suitable for belt conveyors.

Merits

1. It offers higher productivity and lower mining cost.
2. Eliminates drilling and blasting thus it is environmental friendly. It avoids the chance of dilution of pay minerals and also offers higher safety.
3. Eliminates crushing system, thus saves cost.
4. Produce higher quantity of minerals
5. Flexibility of its operation with both dumpers and conveyors
6. Large-scale cutting area and low advancing speed of KSM 2000 makes it possible to load dumpers more conveniently.
7. The cutting drum of KSM 2000 is in the front. Its diameter is large (4.8m) and speed is low (6 rpm). Hence the cutting elements of the drum run at a slower speed and get more time to cool. This phenomenon increases the life of the cutting tools.
8. Due to bigger drum diameter the KSM 2000 can go for larger depth of cut (2.9 m) and more production in comparison with other surface miners. KSM 2000 has more number of cutting elements (540), much more power (total 3200 kW) and more weight (540 t). As a result, it can offer much more cutting force and is able to operate in much tougher conditions in comparison with other surface miners.
9. Cutting can be carried out in any direction. Its operation also offers better slope stability.
10. Milling type of surface miners produce more fines (70% of output are around minus 35 mm), whereas output size from KSM 2000 remains within 75 to 100 mm. This assures better price realization of coal produced by KSM2000 (Table 3.6).

Table 3.6 Brief specification of KRUPP -KSM2000

Name of the Company, Model	KRUPP KSM2000
Operating mass, t	540
No of buckets and cutting picks	4x20 No buckets and 540 cutting picks
Cutting depth, m	0-2.9
Diameter of head, m	4.86
Cutting width, m	7.1
Propel speed, m/min	0.37-3.0
Design production m^3/h	2500
Power, kW	3200

The Milling Type Surface Continuous Miner and its Operation

It is a crawler-mounted machine having a cutting drum centrally mounted beneath the machine at the front end of the same. The cutting drum has helix vanes over which tungsten carbide tipped cutting bits are mounted. A shield (cowl) is fitted behind the cutting drum. By virtue of rotation of the cutting drum the material is cut, crushed between the helix and cowl and thereafter they are conveyed from the two sides of the drum to the centre of the same, from which the material is passed through an intake channel in the shield (cowl) into the primary conveyor (loading conveyor) behind it.

Thereafter the material in turn transfers onto a discharge belt conveyor which is slewable 90° or more on it's either side (left and right). The height of the discharge conveyor is also adjustable. The cutting drum is operated by pole-changing squirrel-cage induction motor and can be raised up, lower down and tilted in the transverse direction by the hydraulic rams to negotiate greater cutting height and dipped coal seam/ore body across the machine. The frame work of the surface continuous miner 300 OSM is of rigid, welded box design with integrated tanks for hydraulic oil diesel and water. The total length without conveyor is approximately 11 m. The machine runs of 3 (3000 SM) or 4 (3800 SM) crawler units. The results in ground pressure which are comparable to other mining machines are 13.8 N/cm^2 and 10.0 N/cm^2 for 3000SM and 3800 SM respectively. Due to the central position of the cutting drum, the rear crawler travels at a lower level than the front crawlers which is equal to the cutting depth- during mining operation. The relatively high traveling speed of the machine permits to change the site rapidly. Its mobile character is emphasized by its independence from external energy source, such as trailing cable. The cutting drums of 3000 mm or 3800 mm width (the cutting drum width determines the machine designation 3000 SM or 3800 SM) are the actual mining units of the machine. The conical picks with tungsten carbide tip inserts presently used on the 3000 SM which rotate in tooth-holders. The tooth-holders are bolted to the outer edge of the drum spiral, which permits the pick flighting to be adopted to the mineral characteristics. The required cutting depth can be adjusted via 2 hydraulic cylinders which are connected to the drum. The surface slope can be regulated within certain limits by laterally tilting the drum to a maximum of 7° (only for 3000 SM), whereas the entyre 3800 SM machine can be tilted. The discharge conveyor of the machine type 3800 SM has a length of 14 m. It can be slewed through an angle of 210°and the discharge height is 8 m.

The latest version of continuous surface miner 4200 SM with an operating weight 180 t and an output of 1200 kW by two engines, has a cutting width of 4.2 m and a maximum cutting depth of 0.6 m. The machine can easily cut and load coal, lignite and overburden at the rate of 3000 TPH. During cutting, the cutting drums strike down to the coal seam/ore body with a speed of more than 6.7 m/s with a very high impact force. Due to these high impact force and cutting speed,the material will be cut and crushed to a more or less uniform in size (size reduction can be controlled by proper selection of the cutting drum depending upon the use of the cut material) which will be very much helpful for not only uniform and efficient loading but also for the free flow of materials through the intake channel. The cutting drum is directly mounted with the main machine body to minimize the shock load. (Fig. 3.23).

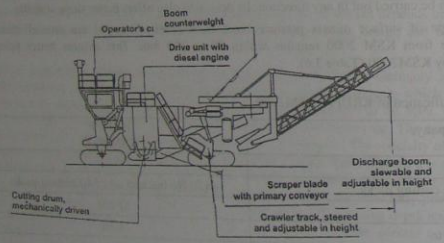

Boom counterweight
Operator's ca
Drive unit with diesel engine
Discharge boom, slewable and adjustable in height
Scraper blade with primary conveyor
Cutting drum, mechanically driven
Crawler track, steered and adjustable in height

Fig. 3.23 : Continuous surface miner (Type 4200 SM).

Advantages to Milling type Surface Continuous Miners

- It eliminates drilling, blasting, etc. which is very good on environmental point of view.
- Primary crushing of material is not needed and can be controlled by the optimum selection of the cutting drum.
- Till date material with an uniaxial compressive strength of 150 MPa can be economically extracted by the latest developed surface continuous miner. .
- Thin coal seam (2.5 cm or less) or thin layer of ore body can be mined nicely and cleanly by the machine which yields better quality ROM and due to this reason beneficiation cost is minimized.

- Since it is a self-mobile machine it is very flexible in operation. The single machine can be used in various section of a mine or many mines.
- It has a very large discharge radius and can load material on dumpers, belt conveyors, mobile conveyor, etc.
- With the help of automatic grade and slope control system correct slope for drainage and mining of dipped seam/ore is possible.
- Since the installation cost is very low and it is a mobile machine, the capital investment is lower.
- More efficient energy utilization is possible by the machine.
- Manpower requirement is very less.
- It has a moderately fast travelling speed.
- It can integrate operations with other mining equipments.
- Performance of beneficiation of coal/ore produced by the system is far better as compared to the other systems.
- Selective mining is possible with the system.

Specification

There are many companies in the world which are manufacturing continuous surface miners, e.g. WRITGEN (SM Series),HURON Manufacturing Co (Easi-Miner)etc. Specification of their few products are given in Table-3.7.

Example of Milling type Surface Continuous Miner

Model-1224 Easi-miner is working in North America for mining lignite and bituminous coal in Texas lignite seam. This unit is loading lignite onto 120t bottom dump trucks at a rate of 3 minutes per truck. In bituminous

Table 3.7 Various types of surface continuous miners coal the production is higher, because the coal fractures more readily.

Name of the Company, Model	Huron Manufacturing Co. EasiMiner Milling type	Writgen 3000 SM	Writgen 3800 SM	Writgen 4200 SM
Operating mass, t	108.84	49.89	117.91	126.98
Dimensions, m	4.9 x 16.5	3.5 x 20.5	7.0 x 25.0	4.9 x 29.9
Cutting depth, m	0.61	0.406	0.610	0.610
Diameter of head, m	2.23	1.22	1.6	1.6
Cutting width, m	4.14	3.05	3.66	4.21
Propel speed, m/min	60.96	21.34	25	27.4
Design production t/h	1633	726-907	1088-1633	1688-2168
Power_kW	895	559	895	1193

Selective mining of multi-thin coal seams (many thin seams having intrusions of small partings in between every seam) in opencast coal benches with the help of continuous surface miners will offer minimizing of coal losses, minimizing of dilution of run-of-mine coal with waste materials, saving in the operational cost and ultimately yield of clean coal. Output from the washeries will also increase and there will be a sharp improvement of heating value of coal. Development of continuous surface miner is further on progress for the recovery of medium hard to harder minerals to eliminate completely need of drilling and blasting along with loading after eliminating the need of primary crushing operations. However, there are some limitations during the use of surface continuous miner. The length of the bench should be as large as possible (say more than 300 m) to reduce the turn around time, to reduce the frequency of loading out bench ends which is generally done by the front end loader and dozer/ripper, etc. It is also very difficult to recover the rib or negotiate the undulation of the seam or ore body although the remote control or infra-red sensing device may be applicable to negotiate the latter to protect the dilution of coal/ore. Although presently best mining performance

achieved by this machine is 1000 TPH but in future with maximizing mining depth and machine speed of more than 10 m/min and with the help of most modern machine a production of more than 6 million cubic meter (bank) per year per machine may be expected (assuming a machine utilization rate of 50%).

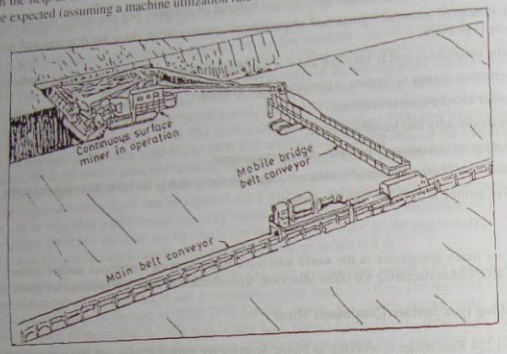


Fig. 3.24 : A Simplified layout of an opencast mine where continuous surface miner is in operation.

ILO SAFETY GUIDANCE DURING WORKING IN THE OPENCAST MINES

Stripping of Overburden

1. Vegetation, such as bushes and trees, must be removed from the overburden before stripping reaches the roots. Foundations of buildings should be similarly removed when stripping reaches their level.
2. When the overburden consists of unconsolidated or blasted material:
   a. excavation by undercutting should not be used;
   b. if mechanical equipment is not used, no place should be worked unless the material is at a stable angle; and
   c. if mechanical equipment is used, no working face should have a vertical height of greater than the maximum height to which the excavation equipment in use can reach.
3. All rock, stone or other material from the stripping of overburden should be dumped or otherwise disposed of in accordance with the provisions of section 10 of this code.
4. Where mining is to be carried out in regions of heavy snowfall or heavy rainfall, particularly in areas liable to flash flooding or where monsoon conditions are prevalent, or in active seismic areas where earthquakes and landslides may be expected, or in areas of volcanic activity, special protective procedures should be developed and copies of these procedures should be sent to the competent authority.

Mining methods

1. Mining methods should be used that will maintain wall, bank and slope stability in places where persons work or travel in performing their assigned tasks. When benching is necessary, the width and height should be based on the type of equipment used for cleaning of benches or for scaling of walls, banks and slopes and the safety of any persons or vehicles working or traveling thereon.
2. Ground conditions that create a hazard to persons should be taken down or supported before other work or travel is permitted in the affected area.

3. Until corrective work is completed, the area should be posted with a warning against entry and when left unattended, a barrier should be installed to impede unauthorized entry.
4. Trimming, scaling and support should be carried out from a safe location.
5. The vertical height of a bench face being worked manually should not exceed 8 m, and where it is being loaded mechanically it should not exceed 20 m, unless – after inspection – the competent authority has given permission in writing for these figures to be exceeded.
6. This provision should not prevent a mine from being worked with two or more benches having a face height not exceeding 8 m and 20 m correspondingly.
7. Persons should not be permitted to work or travel between machinery or equipment and the high wall or bank where the machinery or equipment may hinder escape from falls or slides from the high wall or bank.
8. No person should be deployed in the determined hazard zone of mechanical equipment unless that equipment is effectively immobilized.
9. In cases where an opencast mine is being worked in the vicinity of any underground workings, no face of the opencast mine should be advanced over the underground workings if danger could be occasioned to persons in the underground workings or in the opencast mine unless the competent authority has been informed beforehand and approval has been obtained for the method of working to be implemented.
10. Blasting should not be performed in underground workings that lie within 30 m vertically below the bottom or 60 m horizontally from the wall of any opencast mine in which work is being carried out, unless consultation has taken place between the managements of both enterprises to devise and implement suitable precautionary measures to ensure safe working.
11. No person should work or climb on top of any surge stockpile in active use to which broken rock is fed from above and from which the rock is withdrawn unless all the following provisions are complied with:
    a. He has been authorized to do so by the manager or his representative
    b. The feed to and from the stockpile has been stopped and locked out;
    c. It has been established that the chute below is not in the discharging position;
    d. He is wearing a safety belt attached to a rope of correct length, securely fixed to an anchorage above him; and he is assisted by another person stationed at a safe vantage point above him.
12. In cases where an opencast mine is being worked in the vicinity of any underground workings, no face of the opencast mine should be advanced over the underground workings if danger could be occasioned to persons in the underground workings or in the opencast mine unless the competent authority has been informed beforehand and approval has been obtained for the method of working to be implemented.
13. No person should be permitted to work on a face, wall or other location in an opencast mine from which a dangerous fall could occur unless he is wearing a safety belt of correct length, securely fixed to an anchorage above him and he is assisted by a competent person.
14. At mines where the vertical height of the face exceeds 3 m and where explosives are used, bench drilling should be carried out from the top of the bench.
15. A working face should not be drilled or otherwise worked in a manner which will create an overhang of the face and where unconsolidated rock is being mined, the face and sides should be rammed to prevent collapse.
16. A face should not be undercut by the excavation of a slot at the toe of the face. This provision should not, however, prevent a tunnel or adit being driven into the face for drainage purposes.
17. At a mine where rock is being sluiced or mined by jets of water or other liquid, no person should be permitted to approach the top of the mine face to within a distance equal to twice the height of the face. When working of the jets is suspended, no person may enter the zone earlier excluded unless that zone has been examined and declared to be safe by a competent person. Before the resumption of jet operation, all persons must again be withdrawn from that zone.
18. The maximum height of a sand working face should not exceed the vertical reach of the excavating equipment working at the face.
19. In cases where the projected total depth of any sand pit excavation will exceed the vertical reach of the excavating equipment to be used, the pit should be worked by a series of benches.

Each bench should have separate loading arrangements and should be of sufficient length and breadth to provide safe working conditions.

20. Sand pit faces should be worked over as large a width as is practicable and at any cessation of operations for a period of time exceeding that of one shift, all pit faces should be sloped to prevent any further slump of sand (sand includes all unconsolidated materials or deposits).

21. Where manual workers are engaged on or ahead of overburden, stripping or mineral-winning phases, there should either be a team of two workers or more working together or any solitary worker should be kept under constant observation at all times.

Various Tables, Appendixes, Annexures and Flow Charts

Various tables, appendix, annexure which are enlisted in the next pages are considered to be the approximate indexes/indices/scales of various parameters/factors which are very essential for planning and designing of any surface mining project and layout. Depending upon the site specific conditions, place to place, country to country some of the indexes/indices/scales may vary a little bit but these indexes/indices/scales may be considered as the most average and even some of them one most minimum one. Some of the indices/scales are based on the prices of the particular years mentioned against them in India. Hence planner, mining consultants, etc. should revise them as per the current trends and make them up dated every year. However planner, mining consultants, academicians, students will get a very good guide line and idea from these Tables, Appendixes and Annexures for planning and designing of any surface mining project and layout.

Table 3.8 Category of surface mines

	Category	Capacity per year (Million Tonne)	Capacity per day (Tonne)
1.	Low Capacity	0 to 1	Max. 4000
2.	Medium Capacity	1 to 5	4000 to 20000
3.	High Capacity	5 and above	20000 and above

NOTE: 1. Dragline shall not be used in low capacity mine. 2. Number of working days per year is considered to be 250 days.

Table 3.9 Very good economic stripping ratio

Mineral/Coal		It is mainly depend upon the grade of ore or coal and also on other factors which are already discussed Best Economic Stripping Ratio (Cubic meter of overburden: Tonne of mineral)		Good Economic Stripping Ratio (Cubic meter of overburden : Tonne of mineral)
(A) Coal	1.	Manual	0 to 1:1	1 to 1.2:1
	2.	Semi Mechanized	0 to 1.5:1	1.5 to 2: 1
	3.	Highly Mechanized	0 to 2:1	2 to 3:1
NOTE: With latest improved technology this stripping ratio may exceed 30:1				
(B) Iron Deposit				
	1.	Manual	0 to 0.8:1	0.8 to 1:1
	2.	Semi Mechanized	0 to 1:1	1 to 1.6:1
	3.	Highly Mechanized	0 to 1.6:1	1.6 to 2:1
NOTE : With high mechanization the stripping ratio may exceed 5.5:1				

Table 3.9 continued

(C) Limestone				
	1.	Manual	0:1	0.1:1
	2.	Semi Mechanized	0:1	0.1:1
	3.	Highly Mechanized	0:1	0.2:1
(D) Bauxite				
	1.	Manual	0 to 1:1	1 to 1.5:1
	2.	Semi Mechanized	0 to 1.5:1	1.5 to 2:1
	3.	Highly Mechanized	0 to 2.5:1	2.5 to 4:1
(E) Copper				
	1.	Manual	0 to 0.8:1	0.8 to 1:1
	2.	Semi Mechanized	0 to 1:1	1 to 1.5:1
	3.	Highly Mechanized	0 to 1.5:1	1.5 to 2:1
NOTE : Depending upon the grade the stripping ratio may exceed 4.5:1 for copper.				

Table 3.10 Top soil removal (thickness 0 to 3m)

	Transporting Distance (m)	Equipment
1.	0 to 200	Use either dozer or Front End loader starting from low production capacity mine and above.
2.	200 to 500	Use scraper. For low production capacity mine it is always better if the scraper machine is hired. For medium and high capacity mine ,it should be permanently deployed. If required Ripper may be deployed.
3.	500 and above	Front End Loader and Rear Dump Dumper combination is the best proposition.

Table 3.11 Some approximate idea about bench dimension and number of benches in opencast mines

For flat deposit dipping from 0^0 To 7^0
Combination-Excavator-Shovel, Dragline, Front End Loader. Transporting Mode-Truck, Dumper, Mobile Crusher with High Angle Conveyor.

	Thickness of Coal/ Mineral (m)	Thickness of Overburden (OB), m (m)	Production Requirement (Tonne/day)	Height of Bench (m)		Minimum width of Benches (m)		Maximum Number of Benches	
				Coal/ Mineral	OB	Coal/ Mineral	OB	Coal/ Mineral	OB
1.	0-12	0-12	0-4000	0-12	0-12	25-40	25-40	1	1
2.	12-20	12-20	4000-10000	6-10	6-10 or 12-20	35-40	35-40 or 40-50	2	2 or 1
3.	12-20	24-45	4000-10000	6-10	12-12.5	35-40	40-55	2	2
4.	12-20	36-60	4000-10000	6-10	18-30	35-40	45-60	2	2
5.	20-30	20-30	10000-20000	10-15	10-15 or 20-30	40-50	45-50 or 45-60	2	2 or 1
6.	20-30	40-60	10000-20000	10-15	20-30	40-50	45-60	2	2
7.	20-30	60-90	10000-20000	10-15	20-30	40-50	45-60	2	2 or 3

NOTE : It would be better if a Dragline is employed for a bench height of 18 m and above.

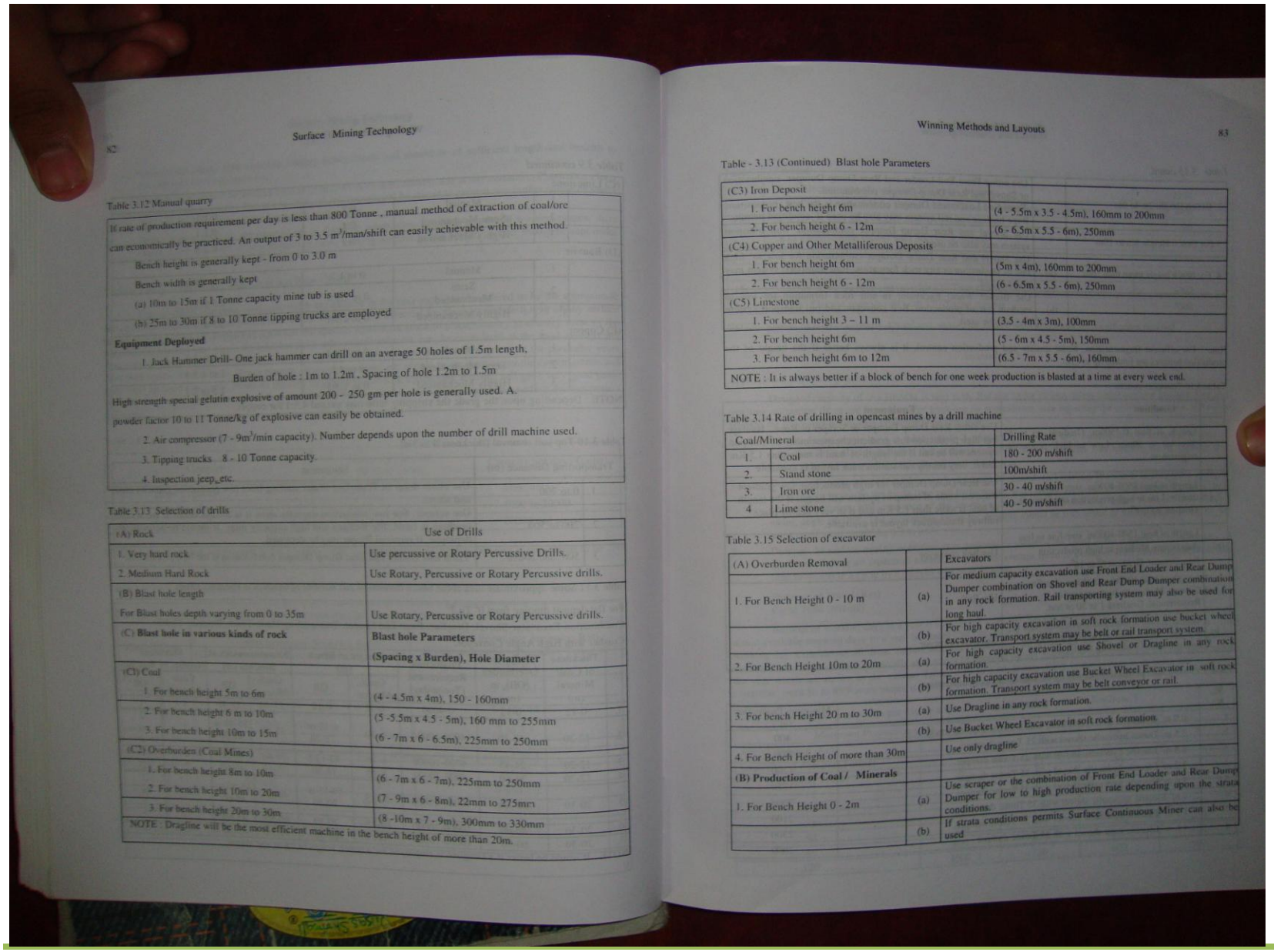


Table 3.12 Manual quarry

If rate of production requirement per day is less than 800 Tonne , manual method of extraction of coal/ore can economically be practiced. An output of 3 to 3.5 m^3/man/shift can easily achievable with this method.
Bench height is generally kept - from 0 to 3.0 m
Bench width is generally kept
(a) 10m to 15m if 1 Tonne capacity mine tub is used
(b) 25m to 30m if 8 to 10 Tonne tipping trucks are employed
Equipment Deployed
1. Jack Hammer Drill- One jack hammer can drill on an average 50 holes of 1.5m length, Burden of hole : 1m to 1.2m , Spacing of hole 1.2m to 1.5m High strength special gelatin explosive of amount 200 - 250 gm per hole is generally used. A powder factor 10 to 11 Tonne/kg of explosive can easily be obtained.
2. Air compressor (7 - 9m^3/min capacity). Number depends upon the number of drill machine used.
3. Tipping trucks 8 - 10 Tonne capacity.
4. Inspection jeep, etc.

Table 3.13 Selection of drills

(A) Rock	Use of Drills
1. Very hard rock	Use percussive or Rotary Percussive Drills.
2. Medium Hard Rock	Use Rotary, Percussive or Rotary Percussive drills.
(B) Blast hole length	
For Blast holes depth varying from 0 to 35m	Use Rotary, Percussive or Rotary Percussive drills.
(C) **Blast hole in various kinds of rock**	**Blast hole Parameters (Spacing x Burden), Hole Diameter**
(C1) Coal	
1. For bench height 5m to 6m	(4 - 4.5m x 4m), 150 - 160mm
2. For bench height 6 m to 10m	(5 -5.5m x 4.5 - 5m), 160 mm to 255mm
3. For bench height 10m to 15m	(6 - 7m x 6 - 6.5m), 225mm to 250mm
(C2) Overburden (Coal Mines)	
1. For bench height 8m to 10m	(6 - 7m x 6 - 7m), 225mm to 250mm
2. For bench height 10m to 20m	(7 - 9m x 6 - 8m), 22mm to 275mm
3. For bench height 20m to 30m	(8 -10m x 7 - 9m), 300mm to 330mm
NOTE : Dragline will be the most efficient machine in the bench height of more than 20m.	

Table - 3.13 (Continued) Blast hole Parameters

(C3) Iron Deposit	
1. For bench height 6m	(4 - 5.5m x 3.5 - 4.5m), 160mm to 200mm
2. For bench height 6 - 12m	(6 - 6.5m x 5.5 - 6m), 250mm
(C4) Copper and Other Metalliferous Deposits	
1. For bench height 6m	(5m x 4m), 160mm to 200mm
2. For bench height 6 - 12m	(6 - 6.5m x 5.5 - 6m), 250mm
(C5) Limestone	
1. For bench height 3 – 11 m	(3.5 - 4m x 3m), 100mm
2. For bench height 6m	(5 - 6m x 4.5 - 5m), 150mm
3. For bench height 6m to 12m	(6.5 - 7m x 5.5 - 6m), 160mm
NOTE : It is always better if a block of bench for one week production is blasted at a time at every week end.	

Table 3.14 Rate of drilling in opencast mines by a drill machine

Coal/Mineral		Drilling Rate
1.	Coal	180 - 200 m/shift
2.	Stand stone	100m/shift
3.	Iron ore	30 - 40 m/shift
4	Lime stone	40 - 50 m/shift

Table 3.15 Selection of excavator

(A) Overburden Removal		Excavators
1. For Bench Height 0 - 10 m	(a)	For medium capacity excavation use Front End Loader and Rear Dump Dumper combination on Shovel and Rear Dump Dumper combination in any rock formation. Rail transporting system may also be used for long haul.
	(b)	For high capacity excavation in soft rock formation use bucket wheel excavator. Transport system may be belt or rail transport system.
2. For Bench Height 10m to 20m	(a)	For high capacity excavation use Shovel or Dragline in any rock formation.
	(b)	For high capacity excavation use Bucket Wheel Excavator in soft rock formation. Transport system may be belt conveyor or rail.
3. For bench Height 20 m to 30m	(a)	Use Dragline in any rock formation.
	(b)	Use Bucket Wheel Excavator in soft rock formation.
4. For Bench Height of more than 30m		Use only dragline
(B) Production of Coal / Minerals		
1. For Bench Height 0 - 2m	(a)	Use scraper or the combination of Front End Loader and Rear Dump Dumper for low to high production rate depending upon the strata conditions.
	(b)	If strata conditions permits Surface Continuous Miner can also be used

Table 3.15 contd.

2. Bench Height 2m to 6m		Use either Front-End-Loader and Rear Dump Dumper combination or Shovel and Rear Dump Dumper combination. Front End Loader and Dumper combination is the best proposition. Rope haulage and rail system may also be used if condition permits.
3 For Bench Height 6m to 10m		Use Shovel and Rear Dump Dumper combination. Rail transport system may also be used.
4. For bench height more than 10m	(a)	Use Shovel and Rear Dump Dumper combination for low to high rate of production in any rock formation. Rail transport system may also be used.
	(b)	Use Bucket Wheel Excavator in soft rock formation for low to medium rate of production. Belt conveyor and rail transport system may also be used.

Note : For efficient fragmentation, better quality and production control, it is always better if the height of the production benches are limited up to 12m.

Table 3.16 Transport system

	Condition	Equipment
1.	Length of haul 0-1500m, Gradient up to 6^0, Size-fine to 50mm, Low to high production requirement	Use either Rear Dump Dumper or Scraper or Rope Haulage. For high production low gradient cheapest mode of transport system will be rail if the length of haul is more than 1.5 Km and if. very nearby rail station/track layout is available.
2. (a)	Length or haul 1500-4000m, size-fine to coarser. Low to high production requirement. Gradient up to 6^0	Use Rear Dump Dumper for high production, low gradient cheapest mode of transport system will be rail if the length of haul is more than 1.5 Km and if in very nearby place railway station/track layout is available.
(b)	Length of haul 1500-4000m, size-fine to less than 60mm. Medium to high production Requirement. Gradient 6^0 to 16^0	Use Belt conveyor
3. (a)	Length of haul 4000m and above ,size-fine to Coarser. Medium to high production Requirement. Gradient-1 in 50 or less.	Use Rail
(b)	Length of haul 4000m and above, Size- fine to less than 60mm, medium to high production Requirement. Gradient up to 16^0.	Use Belt conveyor

Table 3.17 Minimum production per day by the shovel-dumper combination

Sl. No.	Combination	Minimum Production (m^3)
1.	0.9 m^3 Hydraulic shovel (electric) with 8 Tonne Tipping trucks	400
2.	2.5 m^3 Diesel hydraulic shovel with 25 Tonne dumper	800
3.	2.5 m^3 Electric hydraulic shovel with 25 Tonne dumper	1200
4.	3.2 m^3 Electric hydraulic shovel with 35 Tonne dumper	1750
5.	3.6 m^3 Electric hydraulic shovel with 35 Tonne dumper	2000
6.	4 m^3 Electric hydraulic shovel with 35 Tonne dumper	2100
7.	4.5 m^3 Electric hydraulic shovel with 35 Tonne dumper	2300
8.	4.6 m^3 Electric rope shovel with 35 Tonne dumper	2000

Table 3.17 contd.

9.	5 m^3 Electric rope shovel with 35 Tonne, dumper	2100
10.	5.5 m^3 Electric hydraulic shovel with 40 Tonne bottom dump coal hauler	3000
11.	6.3 m^3 Electric rope shovel with 50 Tonne dumper	2700
12.	8 m^3 Electric rope shovel with 50 Tonne dumper	3300
13.	8.3 m^3 Hydraulic shovel with 50 Tonne dumper	4300
14.	10m ^{3} Electric rope shovel with 85 Tonne dumper	4500
15.	12.5 m^3 Electric rope shovel with 120 Tonne dumper	6000
16.	20 m^3 Electric rope shovel with 170 Tonne dumper	9000

NOTE : Shovel of bucket capacity 4.5 m^3 and above can also be used efficiently for overburden removal

Annexure- 3.1

Approximate minimum production rate by various heavy earth moving machineries

Designed capacity of any HEMM is very high. But due to the various delays actual machine available time and machine utilization times are very less. For knowing an approximate idea regarding the cycle of operation of various HEMM the following norms may confidently be followed:

1. Shovel- For a single pass it takes nearly 2 minutes 30 seconds to 3 minutes time to complete a cycle of operation. It is always desirable that a dumper shall be filled up within 3 to 5 passes. Hence capacity of a dumper shall be selected accordingly to match the system.
2. Dragline- For a single pass it takes nearly 1 minute 20 seconds to 1 minutes 30 seconds to complete a cycle. As swing angle increases the cycle time will be increased accordingly. From one sitting position ,the swing angle upto 90^0 accounts 50% clearance of the blasted rock whereas the remaining blasted rocks are cleared upt a swing angle of 180^0. The availability factor for dragline should be 0.95 where as the utilization factor for the same equipment should be 0.9 to 0.93.
3. Dumper- In surface mining operation, it is desirable to maintain the speed of the dumpers in between 16 - 25 Km/Hour. Taking all aspects of normal delays, if the average speed of a dumper is 20 Km/Hour, the maximum time to complete a cycle by a dumper will be as follows:

Distance of Travel (Km)	1	2	3	4
Cycle Time (minute)	15	20	25	30

Number of working days in a year is considered to be 300. But due to various natural calamities and reason, the actual available working days in a year is around 250. Beside this various delays are faced by each equipment in every working day. Considering all these factors and also considering two production and one maintenance shift a minimum consistent rate of production practically very easily be obtained by any machine or combination of machines throughout the year which are shown in the various Tables 3.17 to 3.20. If everything in the system is in order and in healthy condition , even 80 to 95% more output is possible by those machines or combination of machines.

Table 3.18 Minimum production per day by a few dumpers (Traveling speed 20 Km/Hour.)

Minimum Production (m^3)				
Travel distance (Km)	1	2	3	4
Capacity of dumper(Tonne)				
8 (Tipping Truck)	60	45	35	30
15	150	120	90	75
25	250	190	150	125
35	400	300	240	200
50	500	375	300	250
85	800	600	480	400
120	1000	750	600	500

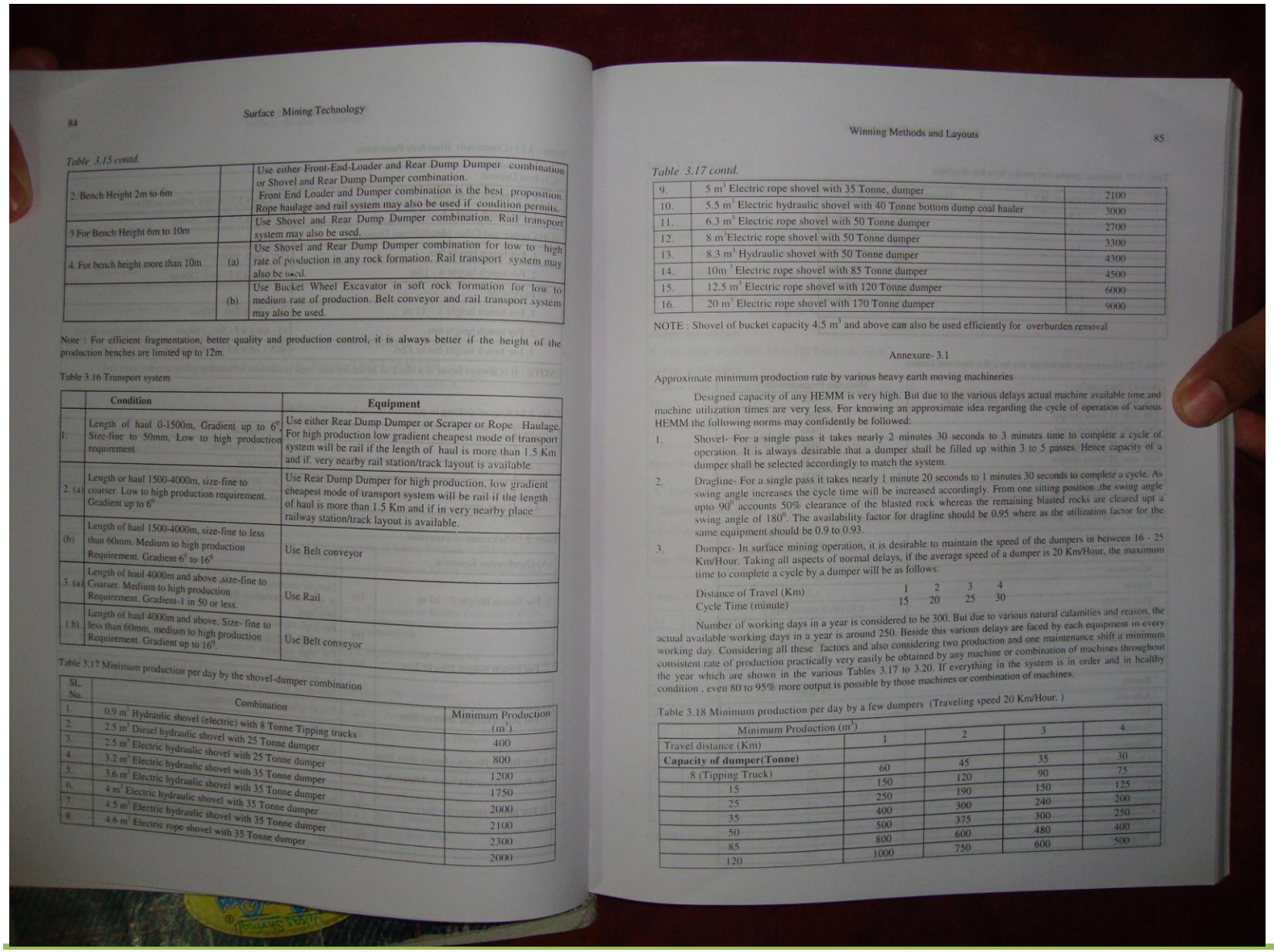


Table 3.15 contd.

2. Bench Height 2m to 6m		Use either Front-End-Loader and Rear Dump Dumper combination or Shovel and Rear Dump Dumper combination. Front End Loader and Dumper combination is the best proposition. Rope haulage and rail system may also be used if condition permits.
3 For Bench Height 6m to 10m		Use Shovel and Rear Dump Dumper combination. Rail transport system may also be used.
4. For bench height more than 10m	(a)	Use Shovel and Rear Dump Dumper combination for low to high rate of production in any rock formation. Rail transport system may also be used.
	(b)	Use Bucket Wheel Excavator in soft rock formation for low to medium rate of production. Belt conveyor and rail transport system may also be used.

Note : For efficient fragmentation, better quality and production control, it is always better if the height of the production benches are limited up to 12m.

Table 3.16 Transport system

	Condition	Equipment
1.	Length of haul 0-1500m, Gradient up to 6^0, Size-fine to 50mm, Low to high production requirement	Use either Rear Dump Dumper or Scraper or Rope Haulage. For high production low gradient cheapest mode of transport system will be rail if the length of haul is more than 1.5 Km and if, very nearby rail station/track layout is available.
2. (a)	Length or haul 1500-4000m, size-fine to coarser. Low to high production requirement. Gradient up to 6^0	Use Rear Dump Dumper for high production, low gradient cheapest mode of transport system will be rail if the length of haul is more than 1.5 Km and if in very nearby place railway station/track layout is available.
(b)	Length of haul 1500-4000m, size-fine to less than 60mm. Medium to high production Requirement. Gradient 6^0 to 16^0	Use Belt conveyor
3. (a)	Length of haul 4000m and above ,size-fine to Coarser. Medium to high production Requirement. Gradient-1 in 50 or less.	Use Rail
(b)	Length of haul 4000m and above, Size- fine to less than 60mm, medium to high production Requirement. Gradient up to 16^0.	Use Belt conveyor

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3.	2.5 m^3 Electric hydraulic shovel with 25 Tonne dumper	1200
4.	3.2 m^3 Electric hydraulic shovel with 35 Tonne dumper	1750
5.	3.6 m^3 Electric hydraulic shovel with 35 Tonne dumper	2000
6.	4 m^3 Electric hydraulic shovel with 35 Tonne dumper	2100
7.	4.5 m^3 Electric hydraulic shovel with 35 Tonne dumper	2300
8.	4.6 m^3 Electric rope shovel with 35 Tonne dumper	2000

Table 3.17 contd.

9.	5 m^3 Electric rope shovel with 35 Tonne, dumper	2100
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12.	8 m^3 Electric rope shovel with 50 Tonne dumper	3300
13.	8.3 m^3 Hydraulic shovel with 50 Tonne dumper	4300
14.	10 m^3 Electric rope shovel with 85 Tonne dumper	4500
15.	12.5 m^3 Electric rope shovel with 120 Tonne dumper	6000
16.	20 m^3 Electric rope shovel with 170 Tonne dumper	9000

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2. Dragline- For a single pass it takes nearly 1 minute 20 seconds to 1 minutes 30 seconds to complete a cycle. As swing angle increases the cycle time will be increased accordingly. From one sitting position ,the swing angle upto 90^0 accounts 50% clearance of the blasted rock whereas the remaining blasted rocks are cleared upt a swing angle of 180^0. The availability factor for dragline should be 0.95 where as the utilization factor for the same equipment should be 0.9 to 0.93.
3. Dumper- In surface mining operation, it is desirable to maintain the speed of the dumpers in between 16 - 25 Km/Hour. Taking all aspects of normal delays, if the average speed of a dumper is 20 Km/Hour, the maximum time to complete a cycle by a dumper will be as follows:

Distance of Travel (Km)	1	2	3	4
Cycle Time (minute)	15	20	25	30

Number of working days in a year is considered to be 300. But due to various natural calamities and reason, the actual available working days in a year is around 250. Beside this various delays are faced by each equipment in every working day. Considering all these factors and also considering two production and one maintenance shift a minimum consistent rate of production practically very easily be obtained by any machine or combination of machines throughout the year which are shown in the various Tables 3.17 to 3.20. If everything in the system is in order and in healthy condition , even 80 to 95% more output is possible by those machines or combination of machines.

Table 3.18 Minimum production per day by a few dumpers (Traveling speed 20 Km/Hour.)

Minimum Production (m^3)				
Travel distance (Km)	1	2	3	4
Capacity of dumper(Tonne)				
8 (Tipping Truck)	60	45	35	30
15	150	120	90	75
25	250	190	150	125
35	400	300	240	200
50	500	375	300	250
85	800	600	480	400
120	1000	750	600	500

Table 3.19 Minimum production per day by a few draglines

Bucket capacity (m^3)	Minimum production (m^3)
3.5	1500
4.0	1700
4.6	1800
6.0	2500
10.0	4000
15.0	8000
24.0	12000
30.6	15000

Table 3.20 Minimum production per day by a few front-end loaders

Bucket Capacity (m^3)	Minimum Production (m^3)
1.52 with 10 Tonne tipping trucks or 15 Tonne dumper	500
2.28 with 25 Tonne dumper	650
3.80 with 25Tonne dumper	1000
4.56 with 35 Tonne dumper	1600
6.84 with 35 Tonne dumper	2000
7.60 with 40 Tonne dumper	2600
9.12 with 50 Tonne dumper	3200

Table 3.21 Approximate specific gravity of some of the common minerals and rocks

Minerals/rocks	Approx. specific gravity
Coal Anthracite	1.5
Bituminous	1.3 to 1.45
Lignite	1.2
Coal (ROM)	0.85 (Bulk density t/m^3)
Sandstone	2.5 to 2.6
Overburden	1.8 to 2.4 (Bulk density t/m^3)
Limestone	2.6
Hematite	5
Bauxite	2.4 to 3.5
Asbestos	2.4
Mica	2.8
Galena	7.5
Magnetite	5.0
Chalcopyrite	4.2
Sphalerite	4.0
Limonite	3.8
Dolomite	2.8
Calcite	2.7
Magnesite	3.0

Table 3.22 Productivity

System combination	O.M.S. in m^3 of Total excavation (coal + overburden)
A) Coal	
1. 3 - 5 m^3 shovel and 35 or 50 Tonne rear dump dumper.	9 to 17
2. 8 m^3 shovel and 50 Tonne rear dump dumper	16 to 18
3. 10 m^3 shovel and 85 or 120 Tonne rear dump dumper	14 to 16
4. 4.6 m^3 rope shovel with 35 or 50 Tonne rear dump dumper and dragline	10 to 28
5. 10m^3 rope shovel with 85 or 120 Tonne rear dump dumper and dragline	27 to 61
6. 20 m^3 rope shovel with 170 Tonne rear dump dumper and Dragline	34 to 49
B) Limestone	
3.3 to 4.6 m^3 shovel and 35 Tonne rear dump dumper	22 to 45
Note :With high mechanization OMS may exceeds 200 Tonne per man-shift	
C) Iron Deposit	
System Combination	
4.6 m^3 shovel and 35 Tonne rear dump dumper	25 to 50
Note :With high mechanization OMS may exceed 250 Tonne per man-shift	

Table - 3.23 Back filling, rehandling of spoil and final surface configuration

			Equipment
A)	Back filling		
	1.	If peak is low	Use either Dozer, Front End Loader or Scraper.
	2.	If peak is moderate	Use either, Dozer, Front End Loader or Scraper
	3.	If peak is very high	Use Dozer
NOTE : Dragline can also be used for backfilling but it can not give excellent configuration.			
B)	Rehandling of Spoil		
	1.	Less than 75%	Use either Dozer, Scraper or Front End Loader
	2.	More than 75%	Use either Dragline, Dozer or Scraper or their combination.
C)	Final Surface Configuration		
	1.	For making flat and smooth surface	Use either Dozer, or Scraper
	2 2.	For making steep and smooth surface	Use Dozer

[NOTE: Dragline may also be used in both the cases (B and C) in selective cases]

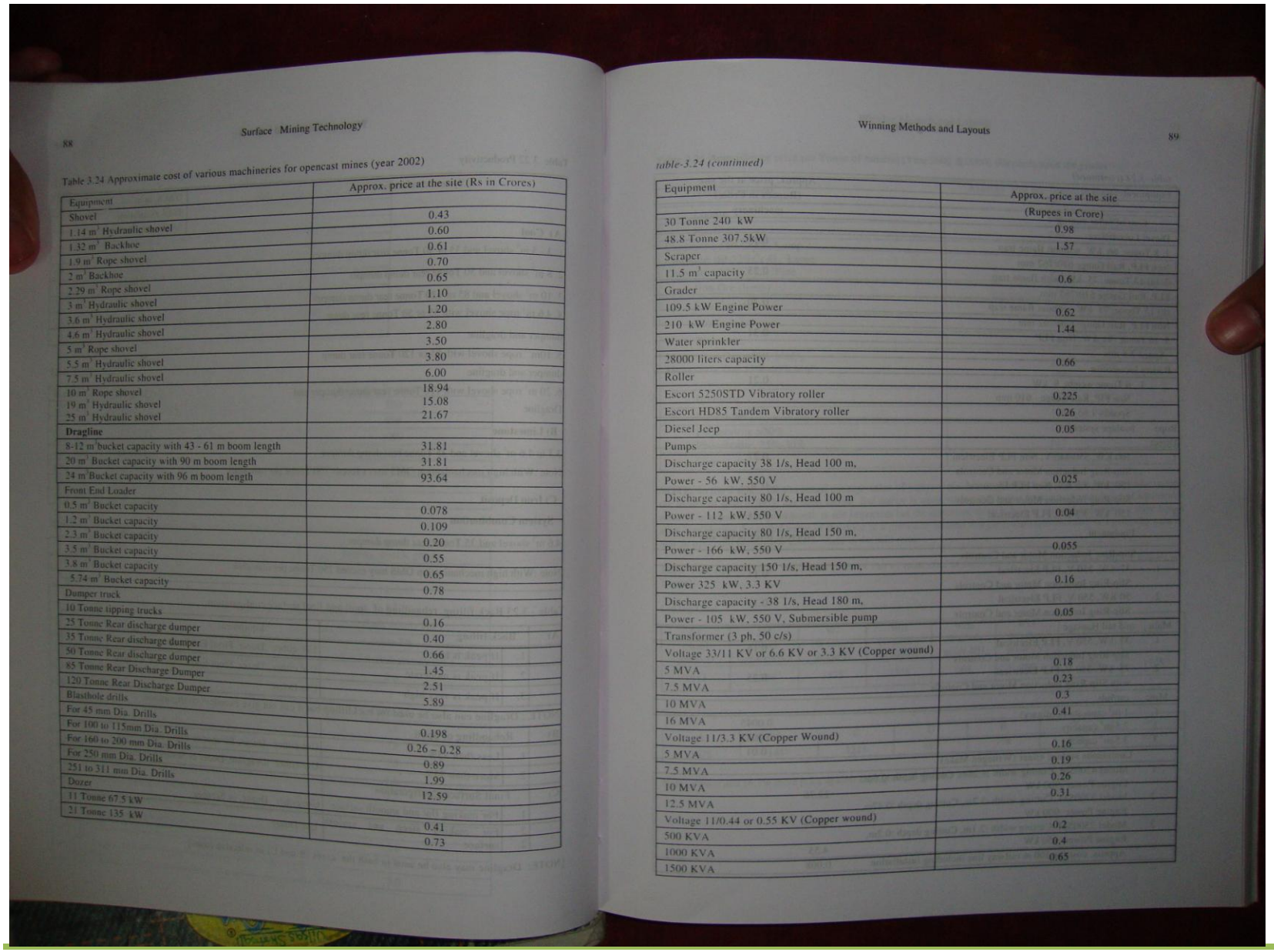


Table 3.24 Approximate cost of various machineries for opencast mines (year 2002)

Equipment	Approx. price at the site (Rs in Crores)
Shovel	
1.14 m^3 Hydraulic shovel	0.43
1.32 m^3 Backhoe	0.60
1.9 m^3 Rope shovel	0.61
2 m^3 Backhoe	0.70
2.29 m^3 Rope shovel	0.65
3 m^3 Hydraulic shovel	1.10
3.6 m^3 Hydraulic shovel	1.20
4.6 m^3 Hydraulic shovel	2.80
5 m^3 Rope shovel	3.50
5.5 m^3 Hydraulic shovel	3.80
7.5 m^3 Hydraulic shovel	6.00
10 m^3 Rope shovel	18.94
19 m^3 Hydraulic shovel	15.08
25 m^3 Hydraulic shovel	21.67
Dragline	
8-12 m^3 bucket capacity with 43 - 61 m boom length	31.81
20 m^3 Bucket capacity with 90 m boom length	31.81
24 m^3 Bucket capacity with 96 m boom length	93.64
Front End Loader	
0.5 m^3 Bucket capacity	0.078
1.2 m^3 Bucket capacity	0.109
2.3 m^3 Bucket capacity	0.20
3.5 m^3 Bucket capacity	0.55
3.8 m^3 Bucket capacity	0.65
5.74 m^3 Bucket capacity	0.78
Dumper truck	
10 Tonne tipping trucks	0.16
25 Tonne Rear discharge dumper	0.40
35 Tonne Rear discharge dumper	0.66
50 Tonne Rear discharge dumper	1.45
85 Tonne Rear Discharge Dumper	2.51
120 Tonne Rear Discharge Dumper	5.89
Blasthole drills	
For 45 mm Dia. Drills	0.198
For 100 to 115mm Dia. Drills	0.26 – 0.28
For 160 to 200 mm Dia. Drills	0.89
For 250 mm Dia. Drills	1.99
251 to 311 mm Dia. Drills	12.59
Dozer	
11 Tonne 67.5 kW	0.41
21 Tonne 135 kW	0.73

table-3.24 (continued)

Equipment	Approx. price at the site (Rupees in Crore)
30 Tonne 240 kW	0.98
48.8 Tonne 307.5kW	1.57
Scraper	
11.5 m^3 capacity	0.6
Grader	
109.5 kW Engine Power	0.62
210 kW Engine Power	1.44
Water sprinkler	
28000 liters capacity	0.66
Roller	
Escort 5250STD Vibratory roller	0.225
Escort HD85 Tandem Vibratory roller	0.26
Diesel Jeep	0.05
Pumps	
Discharge capacity 38 1/s, Head 100 m,	
Power - 56 kW, 550 V	0.025
Discharge capacity 80 1/s, Head 100 m	
Power - 112 kW, 550 V	0.04
Discharge capacity 80 1/s, Head 150 m,	
Power - 166 kW, 550 V	0.055
Discharge capacity 150 1/s, Head 150 m,	
Power 325 kW, 3.3 KV	0.16
Discharge capacity - 38 1/s, Head 180 m,	
Power - 105 kW, 550 V, Submersible pump	0.05
Transformer (3 ph, 50 c/s)	
Voltage 33/11 KV or 6.6 KV or 3.3 KV (Copper wound)	
5 MVA	0.18
7.5 MVA	0.23
10 MVA	0.3
16 MVA	0.41
Voltage 11/3.3 KV (Copper Wound)	
5 MVA	0.16
7.5 MVA	0.19
10 MVA	0.26
12.5 MVA	0.31
Voltage 11/0.44 or 0.55 KV (Copper wound)	
500 KVA	0.2
1000 KVA	0.4
1500 KVA	0.65

By:- Vikas shrimali & Vipul bansal

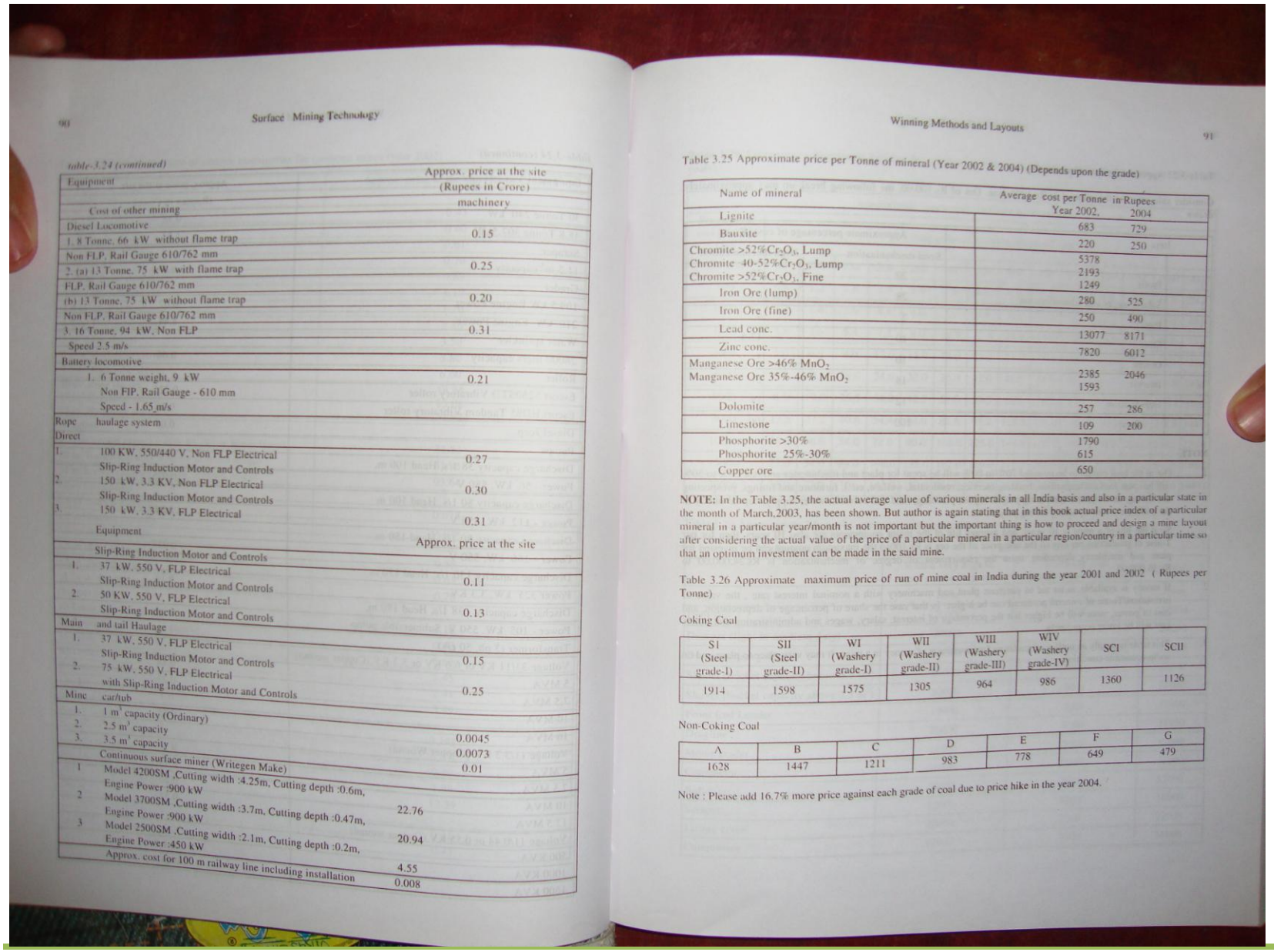


Table-3.24 (continued)

Equipment	Approx. price at the site (Rupees in Crore)
Cost of other mining machinery	
Diesel Locomotive	
1. 8 Tonne, 66 kW without flame trap	0.15
Non FLP, Rail Gauge 610/762 mm	
2. (a) 13 Tonne, 75 kW with flame trap	0.25
FLP, Rail Gauge 610/762 mm	
(b) 13 Tonne, 75 kW without flame trap	0.20
Non FLP, Rail Gauge 610/762 mm	
3. 16 Tonne, 94 kW, Non FLP	0.31
Speed 2.5 m/s	
Battery locomotive	
1. 6 Tonne weight, 9 kW	0.21
Non FIP, Rail Gauge - 610 mm	
Speed - 1.65 m/s	
Rope haulage system	
Direct	
1. 100 KW, 550/440 V, Non FLP Electrical	0.27
Slip-Ring Induction Motor and Controls	
2. 150 kW, 3.3 KV, Non FLP Electrical	0.30
Slip-Ring Induction Motor and Controls	
3. 150 kW, 3.3 KV, FLP Electrical	0.31
Equipment	Approx. price at the site
Slip-Ring Induction Motor and Controls	
1. 37 kW, 550 V, FLP Electrical	0.11
Slip-Ring Induction Motor and Controls	
2. 50 KW, 550 V, FLP Electrical	0.13
Slip-Ring Induction Motor and Controls	
Main and tail Haulage	
1. 37 kW, 550 V, FLP Electrical	0.15
Slip-Ring Induction Motor and Controls	
2. 75 kW, 550 V, FLP Electrical	0.25
with Slip-Ring Induction Motor and Controls	
Mine car/tub	
1. 1 m^3 capacity (Ordinary)	0.0045
2. 2.5 m^3 capacity	0.0073
3. 3.5 m^3 capacity	0.01
Continuous surface miner (Writegen Make)	
1 Model 4200SM ,Cutting width :4.25m, Cutting depth :0.6m, Engine Power :900 kW	22.76
2 Model 3700SM ,Cutting width :3.7m, Cutting depth :0.47m, Engine Power :900 kW	20.94
3 Model 2500SM ,Cutting width :2.1m, Cutting depth :0.2m, Engine Power :450 kW	4.55
Approx. cost for 100 m railway line including installation	0.008

Table 3.25 Approximate price per Tonne of mineral (Year 2002 & 2004) (Depends upon the grade)

Name of mineral	Average cost per Tonne in Rupees Year 2002,	2004
Lignite	683	729
Bauxite	220	250
Chromite >52%Cr_2O_3, Lump	5378	
Chromite 40-52%Cr_2O_3, Lump	2193	
Chromite >52%Cr_2O_3, Fine	1249	
Iron Ore (lump)	280	525
Iron Ore (fine)	250	490
Lead conc.	13077	8171
Zinc conc.	7820	6012
Manganese Ore >46% MnO_2	2385	2046
Manganese Ore 35%-46% MnO_2	1593	
Dolomite	257	286
Limestone	109	200
Phosphorite >30%	1790	
Phosphorite 25%-30%	615	
Copper ore	650	

NOTE: In the Table 3.25, the actual average value of various minerals in all India basis and also in a particular state in the month of March,2003, has been shown. But author is again stating that in this book actual price index of a particular mineral in a particular year/month is not important but the important thing is how to proceed and design a mine layout after considering the actual value of the price of a particular mineral in a particular region/country in a particular time so that an optimum investment can be made in the said mine.

Table 3.26 Approximate maximum price of run of mine coal in India during the year 2001 and 2002 (Rupees per Tonne)

Coking Coal

SI (Steel grade-I)	SII (Steel grade-II)	WI (Washery grade-I)	WII (Washery grade-II)	WIII (Washery grade-III)	WIV (Washery grade-IV)	SCI	SCII
1914	1598	1575	1305	964	986	1360	1126

Non-Coking Coal

A	B	C	D	E	F	G
1628	1447	1211	983	778	649	479

Note : Please add 16.7% more price against each grade of coal due to price hike in the year 2004.

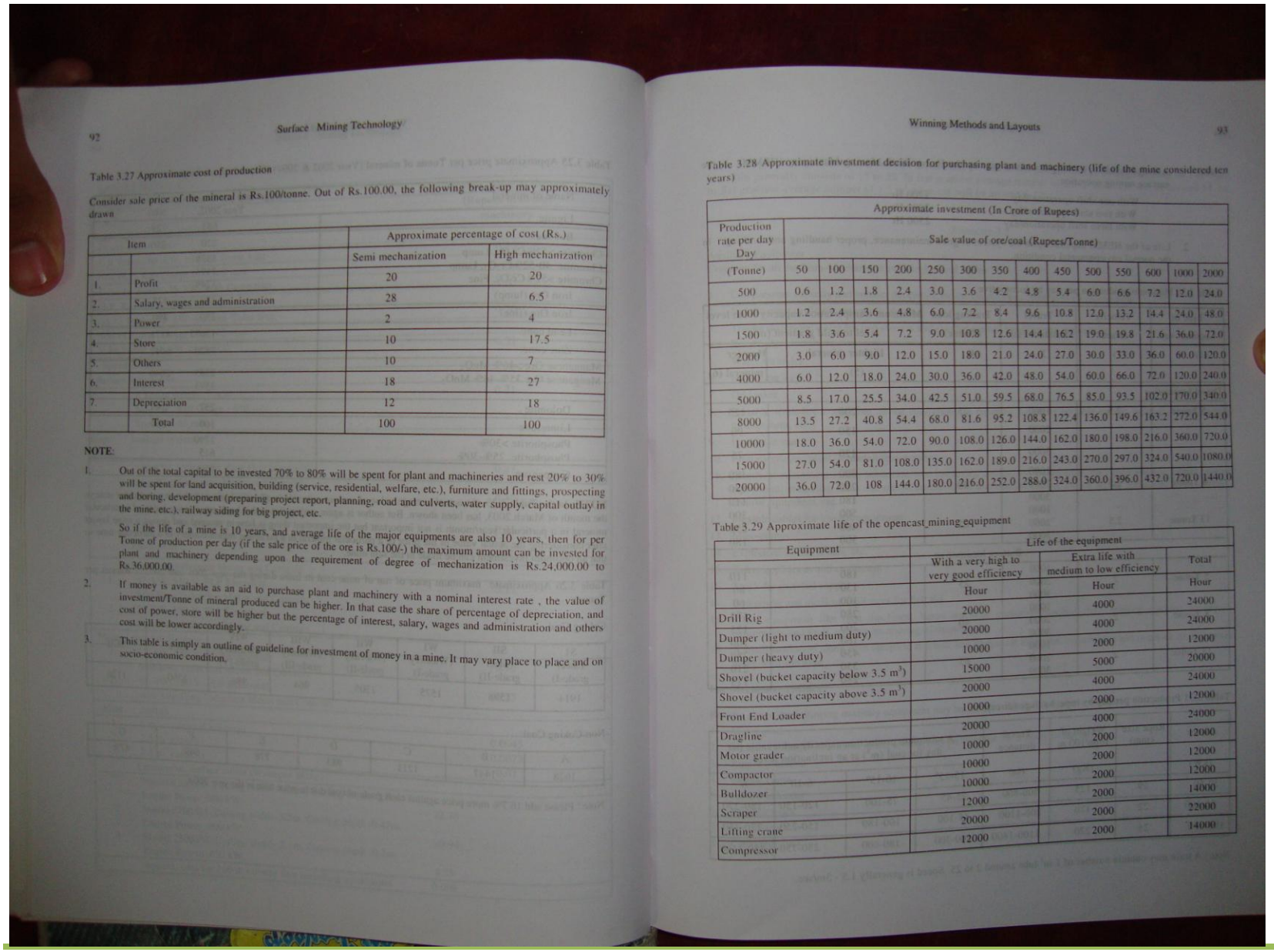


Table 3.27 Approximate cost of production

Consider sale price of the mineral is Rs.100/tonne. Out of Rs.100.00, the following break-up may approximately drawn

Item		Approximate percentage of cost (Rs.)	
		Semi mechanization	High mechanization
1.	Profit	20	20
2.	Salary, wages and administration	28	6.5
3.	Power	2	4
4.	Store	10	17.5
5.	Others	10	7
6.	Interest	18	27
7.	Depreciation	12	18
	Total	100	100

NOTE:

1. Out of the total capital to be invested 70% to 80% will be spent for plant and machineries and rest 20% to 30% will be spent for land acquisition, building (service, residential, welfare, etc.), furniture and fittings, prospecting and boring, development (preparing project report, planning, road and culverts, water supply, capital outlay in the mine, etc.), railway siding for big project, etc.

   So if the life of a mine is 10 years, and average life of the major equipments are also 10 years, then for per Tonne of production per day (if the sale price of the ore is Rs.100/-) the maximum amount can be invested for plant and machinery depending upon the requirement of degree of mechanization is Rs.24,000.00 to Rs.36,000.00

2. If money is available as an aid to purchase plant and machinery with a nominal interest rate , the value of investment/Tonne of mineral produced can be higher. In that case the share of percentage of depreciation, and cost of power, store will be higher but the percentage of interest, salary, wages and administration and others cost will be lower accordingly.

3. This table is simply an outline of guideline for investment of money in a mine. It may vary place to place and on socio-economic condition.

Table 3.28 Approximate investment decision for purchasing plant and machinery (life of the mine considered ten years)

Production rate per day Day	Approximate investment (In Crore of Rupees) Sale value of ore/coal (Rupees/Tonne)													
(Tonne)	50	100	150	200	250	300	350	400	450	500	550	600	1000	2000
500	0.6	1.2	1.8	2.4	3.0	3.6	4.2	4.8	5.4	6.0	6.6	7.2	12.0	24.0
1000	1.2	2.4	3.6	4.8	6.0	7.2	8.4	9.6	10.8	12.0	13.2	14.4	24.0	48.0
1500	1.8	3.6	5.4	7.2	9.0	10.8	12.6	14.4	16.2	19.0	19.8	21.6	36.0	72.0
2000	3.0	6.0	9.0	12.0	15.0	18.0	21.0	24.0	27.0	30.0	33.0	36.0	60.0	120.0
4000	6.0	12.0	18.0	24.0	30.0	36.0	42.0	48.0	54.0	60.0	66.0	72.0	120.0	240.0
5000	8.5	17.0	25.5	34.0	42.5	51.0	59.5	68.0	76.5	85.0	93.5	102.0	170.0	340.0
8000	13.5	27.2	40.8	54.4	68.0	81.6	95.2	108.8	122.4	136.0	149.6	163.2	272.0	544.0
10000	18.0	36.0	54.0	72.0	90.0	108.0	126.0	144.0	162.0	180.0	198.0	216.0	360.0	720.0
15000	27.0	54.0	81.0	108.0	135.0	162.0	189.0	216.0	243.0	270.0	297.0	324.0	540.0	1080.0
20000	36.0	72.0	108	144.0	180.0	216.0	252.0	288.0	324.0	360.0	396.0	432.0	720.0	1440.0

Table 3.29 Approximate life of the opencast mining equipment

Equipment	Life of the equipment		
	With a very high to very good efficiency	Extra life with medium to low efficiency	Total
	Hour	Hour	Hour
Drill Rig	20000	4000	24000
Dumper (light to medium duty)	20000	4000	24000
Dumper (heavy duty)	10000	2000	12000
Shovel (bucket capacity below 3.5 m^3)	15000	5000	20000
Shovel (bucket capacity above 3.5 m^3)	20000	4000	24000
Front End Loader	10000	2000	12000
Dragline	20000	4000	24000
Motor grader	10000	2000	12000
Compactor	10000	2000	12000
Bulldozer	10000	2000	12000
Scraper	12000	2000	14000
Lifting crane	20000	2000	22000
Compressor	12000	2000	14000

Note : 1. Average standard working hour in a year by the Heavy Earth Moving Machinery (HEMM) in the surface mining operation

With one shift operation/day	1200 Hr
With two shift operation/day	2000 Hr
With three shift operation/day	2500 Hr

2. Life of the HEMM depends upon the system of good maintenance, proper handling and working in the normal environmental condition.

Table 3.30 Minimum production per day by locomotive

Type of Locomotives	Velocity m/sec	Travel Distance (m)	Trailing load (Tonne)	Minimum transportation capacity from level level to mild gradient (m^3)	
(1)	(2)	(3)	(4)	Lighter mineral (5)	Heavier mineral (6)
Diesel locomotive					
5 Tonne	2.5	1000	25	200	125
		2000		150	90
		3000		120	75
8 Tonne	2.5	1000	40	300	180
		2000		220	130
		3000		180	110
13 Tonne	2.5	1000	65	500	300
		2000		400	250
		3000		300	180
Battery locomotive					
5 Tonne	2	1000	25	180	110
		2000		130	80
		3000		100	60
8 Tonne	2	1000	40	280	175
		2000		200	125
		3000		160	100
13 Tonne	2	1000	65	450	280
		2000		350	210
		3000		280	175

Table 3.31 Production per day by rope haulage (direct)

Motor (kW)	Rope size (mm)	Weight of rope/100 m (Kg)	Travel distance (m)	A minimum production capacity achievable per day for coal (m^3) at an inclination level			
				15-25°	10-15°	6-10°	6°
30-50	19	125	500-800	25-45	75-100	120-150	180-270
50-100	22	170	800-1100	45-100	100-180	150-250	270-450
100-150	25	220	1100-1400	100-300	180-600	250-750	450-1100

Note : A train may contain number of 1 m^3 tubs around 2 to 25. Speed is generally 1.5 - 3m/sec.

Higher the gradient lesser will be the number of tub. In flat gradient on an average number of 1 m^3 tubs in a train generally consists of 15 to 25. In flat gradient average number of 2.5 m^3 mine car will be around 14-16. In flat gradient average number of 3.5 m^3 mine car will be around 8-12.

Annexure-3.2

Auxiliary Equipment
Common for all the Semi-Mechanized and Mechanized Mines

1. Compressors-capacity and number depends upon the quantity of compressed air requirement.
2. Jack hammer drills-for secondary blasting
3. Tractor dozer-one dozer for two shovels and also for top soil handling and reclamation.
4. Road roller
5. Water sprinkler
6. Inspection jeep
7. Oil tanker
8. Portable workshop containing all the equipment for repairing including portable electric welding set, gas cutting arrangement, etc.
9. Explosive van
10. Primary crusher plant for metalliferous mines.
11. Electrical equipment in the substation like insulators, conductors, transformer, oil circuit breaker, carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, etc.
12. Flexible cables for various equipment.
13. Mine pumps
14. Exploders, circuit testers, etc.
15. Safety appliances and equipment
16. Truck for material supply
17. Equipment and medical appliances for first aid .
18. Office equipment like type machines, Xerox machine, cyclostyle machine, blue printing machine, surveyor's equipments and tools, various monitoring equipment, etc.
19. A small workshop along with complete machine and equipment for repair, maintenance and overhaul.20. Computer along with software

For Mechanized Mines

Besides above the following auxiliary equipment may be incorporated in this category of mines

1. Motor grader
2. Scraper-In semi-mechanized mines scraper may be hired for the removal of top soil.
3. Ripper if needed
4. Mobile crane
5. Front end loader
6. Welding truck
7. Giraffe truck
8. Cable reel truck

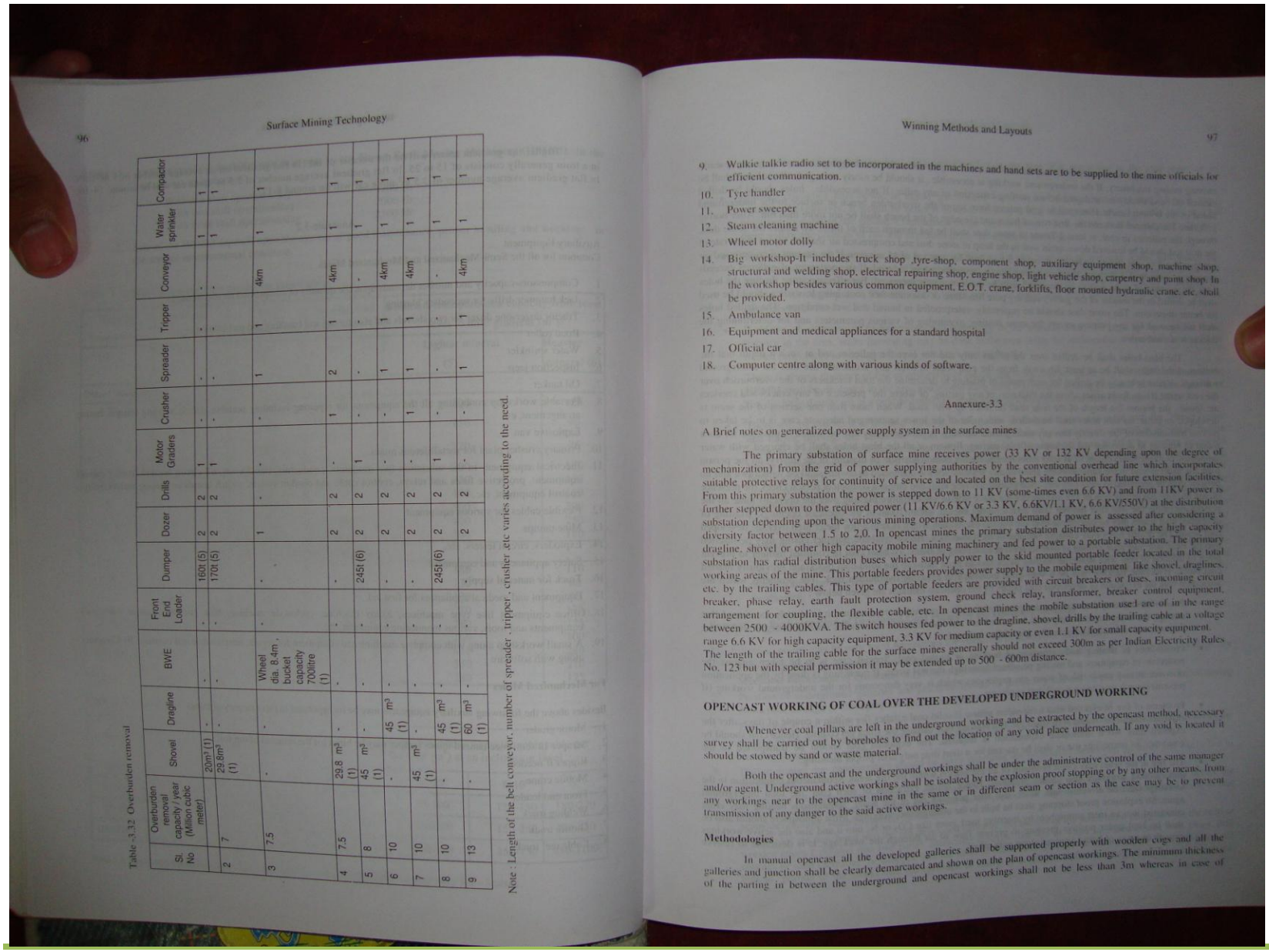


Table -3.32 Overburden removal

Sl. No	Overburden removal capacity / year (Million cubic meter)	Shovel	Dragline	BWE	Front End Loader	Dumper	Dozer	Drills	Motor Graders	Crusher	Spreader	Tripper	Conveyor	Water sprinkler	Compactor
1	-	20m³ (1)	-	-	-	160t (5)	2	2	1	-	-	-	-	1	1
		29.8m³ (1)	-	-	-	170t (5)	2	2	1	-	-	-	-	1	1
2	7														
3	7.5	-	-	Wheel dia. 8.4m , bucket capacity 700litre (1)	-	-	1	-	-	-	1	1	4km	1	1
4	7.5	29.8 m³ (1)	-	-	-	-	2	2	-	1	2	1	4km	1	1
5	8	45 m³ (1)	-	-	-	245t (6)	2	2	1	-	-	-	-	1	1
6	10	-	45 m³ (1)	-	-	-	2	2	-	-	1	1	4km	1	1
7	10	45 m³ (1)	-	-	-	-	2	2	-	1	1	1	4km	1	1
8	10	-	45 m³ (1)	-	-	245t (6)	2	2	1	-	-	-	-	1	1
9	13	-	60 m³ (1)	-	-	-	2	2	-	-	1	1	4km	1	1

Note : Length of the belt conveyor, number of spreader , tripper , crusher ,etc varies according to the need.

9. Walkie talkie radio set to be incorporated in the machines and hand sets are to be supplied to the mine officials for efficient communication.
10. Tyre handler
11. Power sweeper
12. Steam cleaning machine
13. Wheel motor dolly
14. Big workshop-It includes truck shop ,tyre-shop, component shop, auxiliary equipment shop, machine shop, structural and welding shop, electrical repairing shop, engine shop, light vehicle shop, carpentry and paint shop. In the workshop besides various common equipment, E.O.T. crane, forklifts, floor mounted hydraulic crane, etc. shall be provided.
15. Ambulance van
16. Equipment and medical appliances for a standard hospital
17. Official car
18. Computer centre along with various kinds of software.

Annexure-3.3

A Brief notes on generalized power supply system in the surface mines

The primary substation of surface mine receives power (33 KV or 132 KV depending upon the degree of mechanization) from the grid of power supplying authorities by the conventional overhead line which incorporates suitable protective relays for continuity of service and located on the best site condition for future extension facilities. From this primary substation the power is stepped down to 11 KV (some-times even 6.6 KV) and from 11KV power is further stepped down to the required power (11 KV/6.6 KV or 3.3 KV, 6.6KV/1.1 KV, 6.6 KV/550V) at the distribution substation depending upon the various mining operations. Maximum demand of power is assessed after considering a diversity factor between 1.5 to 2.0. In opencast mines the primary substation distributes power to the high capacity dragline, shovel or other high capacity mobile mining machinery and fed power to a portable substation. The primary substation has radial distribution buses which supply power to the skid mounted portable feeder located in the total working areas of the mine. This portable feeders provides power supply to the mobile equipment like shovel, draglines, etc. by the trailing cables. This type of portable feeders are provided with circuit breakers or fuses, incoming circuit breaker, phase relay, earth fault protection system, ground check relay, transformer, breaker control equipment, arrangement for coupling, the flexible cable, etc. In opencast mines the mobile substation use1 are of in the range between 2500 - 4000KVA. The switch houses fed power to the dragline, shovel, drills by the trailing cable at a voltage range 6.6 KV for high capacity equipment, 3.3 KV for medium capacity or even 1.1 KV for small capacity equipment. The length of the trailing cable for the surface mines generally should not exceed 300m as per Indian Electricity Rules No. 123 but with special permission it may be extended up to 500 - 600m distance.

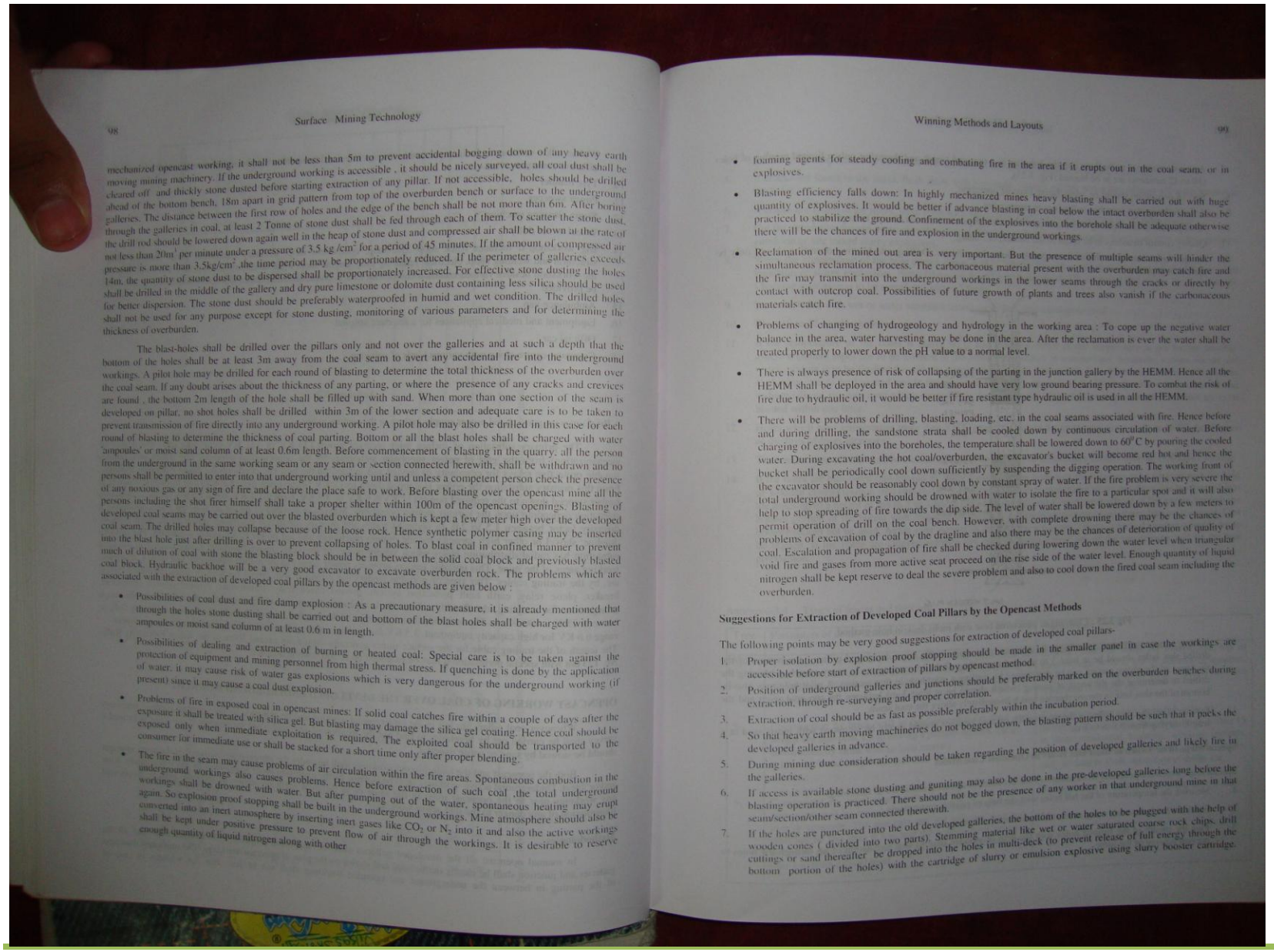
OPENCAST WORKING OF COAL OVER THE DEVELOPED UNDERGROUND WORKING

Whenever coal pillars are left in the underground working and be extracted by the opencast method, necessary survey shall be carried out by boreholes to find out the location of any void place underneath. If any void is located it should be stowed by sand or waste material.

Both the opencast and the underground workings shall be under the administrative control of the same manager and/or agent. Underground active workings shall be isolated by the explosion proof stopping or by any other means, from any workings near to the opencast mine in the same or in different seam or section as the case may be to prevent transmission of any danger to the said active workings.

Methodologies

In manual opencast all the developed galleries shall be supported properly with wooden cogs and all the galleries and junction shall be clearly demarcated and shown on the plan of opencast workings. The minimum thickness of the parting in between the underground and opencast workings shall not be less than 3m whereas in case of



mechanized opencast working, it shall not be less than 5m to prevent accidental bogging down of any heavy earth moving mining machinery. If the underground working is accessible , it should be nicely surveyed, all coal dust shall be cleared off and thickly stone dusted before starting extraction of any pillar. If not accessible, holes should be drilled ahead of the bottom bench, 18m apart in grid pattern from top of the overburden bench or surface to the underground galleries. The distance between the first row of holes and the edge of the bench shall be not more than 6m. After boring through the galleries in coal, at least 2 Tonne of stone dust shall be fed through each of them. To scatter the stone dust, the drill rod should be lowered down again well in the heap of stone dust and compressed air shall be blown at the rate of not less than 20m^3 per minute under a pressure of 3.5 kg /cm^2 for a period of 45 minutes. If the amount of compressed air pressure is more than 3.5kg/cm^2 ,the time period may be proportionately reduced. If the perimeter of galleries exceeds 14m, the quantity of stone dust to be dispersed shall be proportionately increased. For effective stone dusting the holes shall be drilled in the middle of the gallery and dry pure limestone or dolomite dust containing less silica should be used for better dispersion. The stone dust should be preferably waterproofed in humid and wet condition. The drilled holes shall not be used for any purpose except for stone dusting, monitoring of various parameters and for determining the thickness of overburden.

The blast-holes shall be drilled over the pillars only and not over the galleries and at such a depth that the bottom of the holes shall be at least 3m away from the coal seam to avert any accidental fire into the underground workings. A pilot hole may be drilled for each round of blasting to determine the total thickness of the overburden over the coal seam. If any doubt arises about the thickness of any parting, or where the presence of any cracks and crevices are found , the bottom 2m length of the hole shall be filled up with sand. When more than one section of the seam is developed on pillar, no shot holes shall be drilled within 3m of the lower section and adequate care is to be taken to prevent transmission of fire directly into any underground working. A pilot hole may also be drilled in this case for each round of blasting to determine the thickness of coal parting. Bottom or all the blast holes shall be charged with water 'ampoules' or moist sand column of at least 0.6m length. Before commencement of blasting in the quarry, all the person from the underground in the same working seam or any seam or section connected herewith, shall be withdrawn and no persons shall be permitted to enter into that underground working until and unless a competent person check the presence of any noxious gas or any sign of fire and declare the place safe to work. Before blasting over the opencast mine all the persons including the shot firer himself shall take a proper shelter within 100m of the opencast openings. Blasting of developed coal seams may be carried out over the blasted overburden which is kept a few meter high over the developed coal seam. The drilled holes may collapse because of the loose rock. Hence synthetic polymer casing may be inserted into the blast hole just after drilling is over to prevent collapsing of holes. To blast coal in confined manner to prevent much of dilution of coal with stone the blasting block should be in between the solid coal block and previously blasted coal block. Hydraulic backhoe will be a very good excavator to excavate overburden rock. The problems which are associated with the extraction of developed coal pillars by the opencast methods are given below :

- Possibilities of coal dust and fire damp explosion : As a precautionary measure, it is already mentioned that through the holes stone dusting shall be carried out and bottom of the blast holes shall be charged with water ampoules or moist sand column of at least 0.6 m in length.
- Possibilities of dealing and extraction of burning or heated coal: Special care is to be taken against the protection of equipment and mining personnel from high thermal stress. If quenching is done by the application of water, it may cause risk of water gas explosions which is very dangerous for the underground working (if present) since it may cause a coal dust explosion.
- Problems of fire in exposed coal in opencast mines: If solid coal catches fire within a couple of days after the exposure it shall be treated with silica gel. But blasting may damage the silica gel coating. Hence coal should be exposed only when immediate exploitation is required. The exploited coal should be transported to the consumer for immediate use or shall be stacked for a short time only after proper blending.
- The fire in the seam may cause problems of air circulation within the fire areas. Spontaneous combustion in the underground workings also causes problems. Hence before extraction of such coal ,the total underground workings shall be drowned with water. But after pumping out of the water, spontaneous heating may erupt again. So explosion proof stopping shall be built in the underground workings. Mine atmosphere should also be converted into an inert atmosphere by inserting inert gases like CO_2 or N_2 into it and also the active workings shall be kept under positive pressure to prevent flow of air through the workings. It is desirable to reserve enough quantity of liquid nitrogen along with other

- foaming agents for steady cooling and combating fire in the area if it erupts out in the coal seam, or in explosives.
- Blasting efficiency falls down: In highly mechanized mines heavy blasting shall be carried out with huge quantity of explosives. It would be better if advance blasting in coal below the intact overburden shall also be practiced to stabilize the ground. Confinement of the explosives into the borehole shall be adequate otherwise there will be the chances of fire and explosion in the underground workings.
- Reclamation of the mined out area is very important. But the presence of multiple seams will hinder the simultaneous reclamation process. The carbonaceous material present with the overburden may catch fire and the fire may transmit into the underground workings in the lower seams through the cracks or directly by contact with outcrop coal. Possibilities of future growth of plants and trees also vanish if the carbonaceous materials catch fire.
- Problems of changing of hydrogeology and hydrology in the working area : To cope up the negative water balance in the area, water harvesting may be done in the area. After the reclamation is ever the water shall be treated properly to lower down the pH value to a normal level.
- There is always presence of risk of collapsing of the parting in the junction gallery by the HEMM. Hence all the HEMM shall be deployed in the area and should have very low ground bearing pressure. To combat the risk of fire due to hydraulic oil, it would be better if fire resistant type hydraulic oil is used in all the HEMM.
- There will be problems of drilling, blasting, loading, etc. in the coal seams associated with fire. Hence before and during drilling, the sandstone strata shall be cooled down by continuous circulation of water. Before charging of explosives into the boreholes, the temperature shall be lowered down to 60°C by pouring the cooled water. During excavating the hot coal/overburden, the excavator's bucket will become red hot and hence the bucket shall be periodically cool down sufficiently by suspending the digging operation. The working front of the excavator should be reasonably cool down by constant spray of water. If the fire problem is very severe the total underground working should be drowned with water to isolate the fire to a particular spot and it will also help to stop spreading of fire towards the dip side. The level of water shall be lowered down by a few meters to permit operation of drill on the coal bench. However, with complete drowning there may be the chances of problems of excavation of coal by the dragline and also there may be the chances of deterioration of quality of coal. Escalation and propagation of fire shall be checked during lowering down the water level when triangular void fire and gases from more active seat proceed on the rise side of the water level. Enough quantity of liquid nitrogen shall be kept reserve to deal the severe problem and also to cool down the fired coal seam including the overburden.

Suggestions for Extraction of Developed Coal Pillars by the Opencast Methods

The following points may be very good suggestions for extraction of developed coal pillars-

1. Proper isolation by explosion proof stopping should be made in the smaller panel in case the workings are accessible before start of extraction of pillars by opencast method.
2. Position of underground galleries and junctions should be preferably marked on the overburden benches during extraction, through re-surveying and proper correlation.
3. Extraction of coal should be as fast as possible preferably within the incubation period.
4. So that heavy earth moving machineries do not bogged down, the blasting pattern should be such that it packs the developed galleries in advance.
5. During mining due consideration should be taken regarding the position of developed galleries and likely fire in the galleries.
6. If access is available stone dusting and guniting may also be done in the pre-developed galleries long before the blasting operation is practiced. There should not be the presence of any worker in that underground mine in that seam/section/other seam connected therewith.
7. If the holes are punctured into the old developed galleries, the bottom of the holes to be plugged with the help of wooden cones (divided into two parts). Stemming material like wet or water saturated coarse rock chips, drill cuttings or sand thereafter be dropped into the holes in multi-deck (to prevent release of full energy through the bottom portion of the holes) with the cartridge of slurry or emulsion explosive using slurry booster cartridge.

Again stemming is to be done into the holes up to a required length and thereafter only a limited number of holes (10 to 12 numbers) are to be blasted (Fig. 3.25).

8. Special precaution should be taken during blasting operation.
9. A fire fighting plan must be chalked out to combat fire.
10. Arrangement of fire fighting with water jets with enough pressure should always be kept ready.
11. Quality control measures to be taken in the method of mining.
12. No shot holes should be drilled immediately above the galleries. Drilling and blasting operation should only be done in coal pillars.

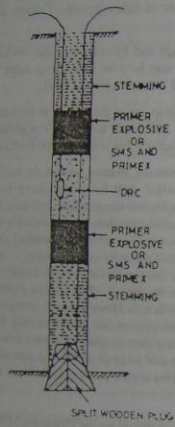


Fig 3.25 : Charging of punctured hole with multi deck in hole method

13. If there is any multi-section workings in the same coal seam in the same developed pillars , the bottom of the drilled shot holes should be at least 3 m above the working of the lower section. In every round of blasting the thickness of the above parting can be determined by drilling a pilot hole in that section. A pilot hole may also be drilled to determine the thickness of parting (which should be at least 3 m) between the top coal seam and the bottom of the shot hole in the overburden bench immediately above the coal seam.
14. Bottom of all the shot holes in the overburden or coal benches should nicely be stemmed with moist sand at a length varying from 0.8 m to 1 m.

Special Steps to be Taken to Blast Hot Holes in Opencast Coal Mines

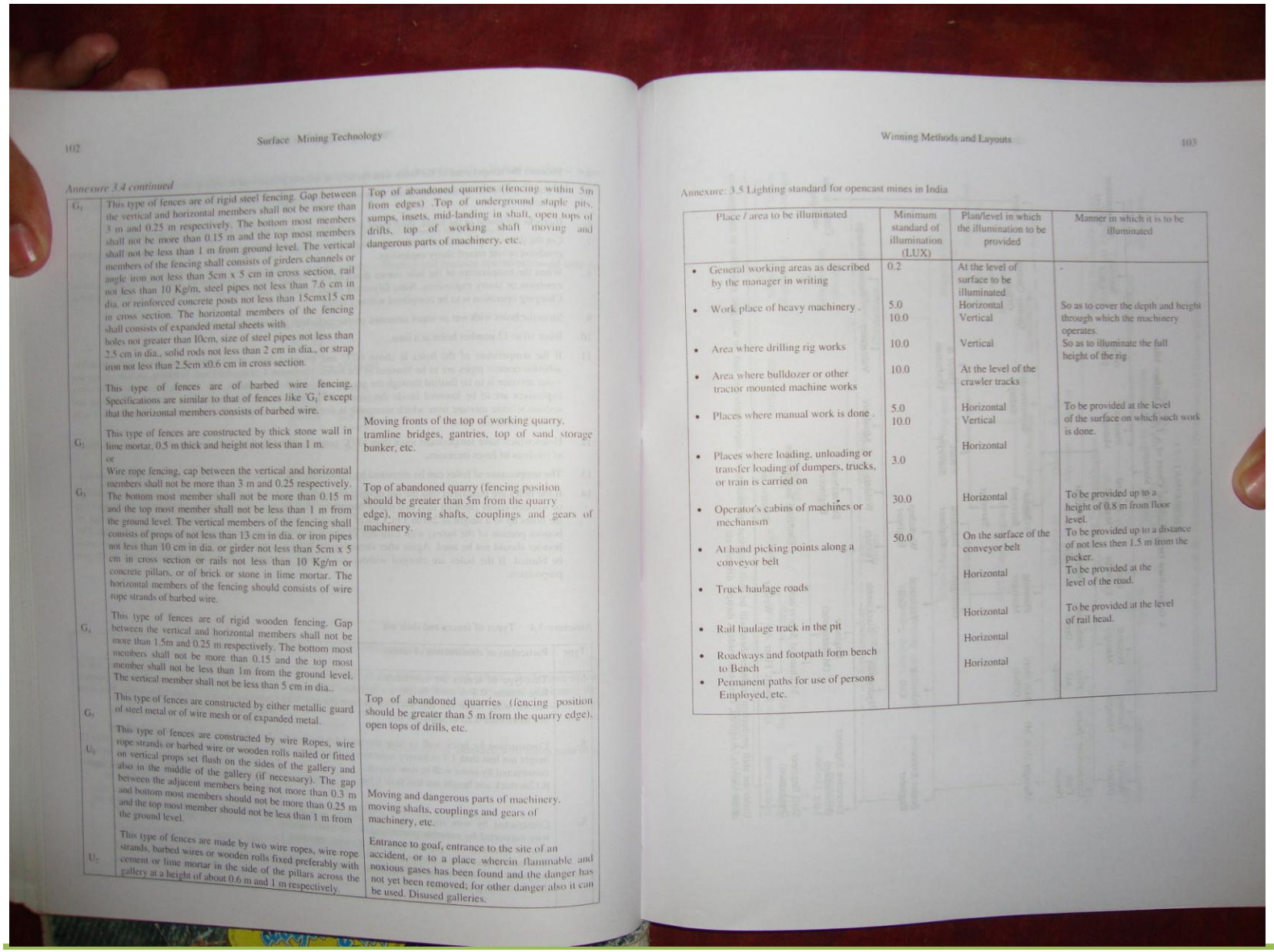
Special steps which are generally to be taken to blast hot holes in opencast mines are as fallowing:

1. Measure the temperature of the hot holes with the help of reliable instruments.
2. Flush the hot holes with water at least 12 hours before blasting.
3. Plug the mouth of the flushed holes with the help of wooden plug to stop circulation of air through the holes to prevent heating of the same.

4. Measure the temperature of the holes with the help of reliable instruments at regular intervals.
5. If cracks are found in the holes, flush them with bentonite, sodium silicate and water mixture till water retains in the holes.
6. If cracks are found in the holes, flush them with water-guar gum (up to 5%) mixture to fill these cracks.
7. Cut the detonating fuse in proper length and pass them through the centre of each charge cartridge of heat resistant emulsion or site mixed slurry explosives.
8. When the temperature of the hole comes down to 60^0C charge the above mentioned holes with the heat resistant emulsion or slurry explosives. Nitro Glycerine based explosive and LOX are prohibited to use in the hot holes. Charging operation is to be completed within 2 hours.
9. Stem the holes with wet or water saturated coarse rock chips drill cuttings or sand.
10. Blast 10 to 12 number holes at a time.
11. If the temperature of the holes is above 60^0C and lowering down of the temperature is not further possible, asbestos cement pipes are to be lowered in the holes (maximum 4 in number) and bentonite, sodium silicate and water mixture is to be flushed through the annulus between the pipe and the hole. Heat resistant emulsion or slurry explosives are to be lowered inside the pipe and the empty space is to be filled up with water, bentonite and sodium silicate mixture over which stemming is done with rock chips, drill cuttings or sand. The holes are to be blasted within one hour.
12. Holes which have temperature more than 100^0C generally should not be blasted since at above 100^0C the chances of misfires of fuses increases.
13. The temperature of holes can be measured by embedding Pt-100 temperature sensors with the explosive cartridge.
14. If the holes are punctured into the old developed galleries, the bottom of the holes are to be plugged with the help of wooden cones (splitted into two parts). Stemming material as mentioned above are thereafter be dropped into the holes up to a height of 2m to 3 m. Charge the holes in multi-deck (to prevent release of full energy through the bottom portion of the holes) with cartridge of slurry or emulsion explosives using slurry booster cartridge. Cast booster should not be used. Again after stemming the holes to a required height ,a limited number of holes are to be blasted. If the holes are charged without any decking arrangement use of the top priming will be the best proposition.

Annexure-3.4 Types of fences and their use

Type	Particulars of construction of fences	Place of use
S_1	This type of fences are constructed by the brick wall in lime mortar, 0.4 m thick, height not less than 1.2 m with a barbed wire, fence not less than 0.6 m high (having wire not more than 0.25 m apart) on top.	Storing tanks of water for mining use, top of sand storage bunker, top of abandoned quarries, subsided areas tramlines lying within 15m from residential building, etc.
S_2	Construction by brick wall in lime mortar 0.4m thick and height not less than 1.5 m having a parapet wall on top or constructed by stone wall in lime mortar, 0.15m thick and height not less than 1.5m	Top of abandoned shaft, top of abandoned quarries, subsided area, tramlines lying within 15 m from residential building, working inclines, tops of underground staple pits, sumps, insets, mid- landing in shaft, open tops of drifts, etc.
S_3	Constructed by wire ropes, wire rope strands or barbed wire supported by movable posts/stands made from wood, iron or concrete. The gap between the adjacent members shall not be less than 0.3 m and the bottom most member not more than 0.25 m and the top most member not less than 1 m from ground level.	Moving fronts of the top of working quarry ,etc.



Annexure 3.4 continued

G_1	This type of fences are of rigid steel fencing. Gap between the vertical and horizontal members shall not be more than 3 m and 0.25 m respectively. The bottom most members shall not be more than 0.15 m and the top most members shall not be less than 1 m from ground level. The vertical members of the fencing shall consists of girders channels or angle iron not less than 5cm x 5 cm in cross section, rail not less than 10 Kg/m, steel pipes not less than 7.6 cm in dia, or reinforced concrete posts not less than 15cmx15 cm in cross section. The horizontal members of the fencing shall consists of expanded metal sheets with holes not greater than 10cm, size of steel pipes not less than 2.5 cm in dia., solid rods not less than 2 cm in dia., or strap iron not less than 2.5cm x0.6 cm in cross section.	Top of abandoned quarries (fencing within 5m from edges) .Top of underground staple pits, sumps, insets, mid-landing in shaft, open tops of drifts, top of working shaft moving and dangerous parts of machinery, etc.
	This type of fences are of barbed wire fencing. Specifications are similar to that of fences like 'G_1' except that the horizontal members consists of barbed wire.	
G_2	This type of fences are constructed by thick stone wall in lime mortar, 0.5 m thick and height not less than 1 m. or Wire rope fencing, cap between the vertical and horizontal members shall not be more than 3 m and 0.25 respectively.	Moving fronts of the top of working quarry, tramline bridges, gantries, top of sand storage bunker, etc.
G_3	The bottom most member shall not be more than 0.15 m and the top most member shall not be less than 1 m from the ground level. The vertical members of the fencing shall consists of props of not less than 13 cm in dia. or iron pipes not less than 10 cm in dia. or girder not less than 5cm x 5 cm in cross section or rails not less than 10 Kg/m or concrete pillars, or of brick or stone in lime mortar. The horizontal members of the fencing should consists of wire rope strands of barbed wire.	Top of abandoned quarry (fencing position should be greater than 5m from the quarry edge), moving shafts, couplings and gears of machinery.
G_4	This type of fences are of rigid wooden fencing. Gap between the vertical and horizontal members shall not be more than 1.5m and 0.25 m respectively. The bottom most members shall not be more than 0.15 and the top most member shall not be less than 1m from the ground level. The vertical member shall not be less than 5 cm in dia..	
G_5	This type of fences are constructed by either metallic guard of steel metal or of wire mesh or of expanded metal.	Top of abandoned quarries (fencing position should be greater than 5 m from the quarry edge), open tops of drills, etc.
U_1	This type of fences are constructed by wire Ropes, wire rope strands or barbed wire or wooden rolls nailed or fitted on vertical props set flush on the sides of the gallery and also in the middle of the gallery (if necessary). The gap between the adjacent members being not more than 0.3 m and bottom most members should not be more than 0.25 m and the top most member should not be less than 1 m from the ground level.	Moving and dangerous parts of machinery, moving shafts, couplings and gears of machinery, etc.
U_2	This type of fences are made by two wire ropes, wire rope strands, barbed wires or wooden rolls fixed preferably with cement or lime mortar in the side of the pillars across the gallery at a height of about 0.6 m and 1 m respectively.	Entrance to goaf, entrance to the site of an accident, or to a place wherein flammable and noxious gases has been found and the danger has not yet been removed; for other danger also it can be used. Disused galleries.

Annexure: 3.5 Lighting standard for opencast mines in India

Place / area to be illuminated	Minimum standard of illumination (LUX)	Plan/level in which the illumination to be provided	Manner in which it is to be illuminated
• General working areas as described by the manager in writing	0.2	At the level of surface to be illuminated	-
• Work place of heavy machinery .	5.0 10.0	Horizontal Vertical	So as to cover the depth and height through which the machinery operates.
• Area where drilling rig works	10.0	Vertical	So as to illuminate the full height of the rig
• Area where bulldozer or other tractor mounted machine works	10.0	At the level of the crawler tracks	
• Places where manual work is done .	5.0 10.0	Horizontal Vertical	To be provided at the level of the surface on which such work is done.
• Places where loading, unloading or transfer loading of dumpers, trucks, or train is carried on	3.0	Horizontal	
• Operator's cabins of machines or mechanism	30.0	Horizontal	To be provided up to a height of 0.8 m from floor level.
• At hand picking points along a conveyor belt	50.0	On the surface of the conveyor belt	To be provided up to a distance of not less then 1.5 m from the picker.
• Truck haulage roads		Horizontal	To be provided at the level of the road.
• Rail haulage track in the pit		Horizontal	To be provided at the level of rail head.
• Roadways and footpath form bench to Bench		Horizontal	
• Permanent paths for use of persons Employed, etc.		Horizontal	

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FLOW CHART - 3.1

A Generalised Organisational Chart of A Large Surface Mine

Under the shift production engineer there will be shift electrical and mechanical supervisors and also he will give guidelines to the operators of the shovels, draglines, front-end loaders, dumper drivers, etc.

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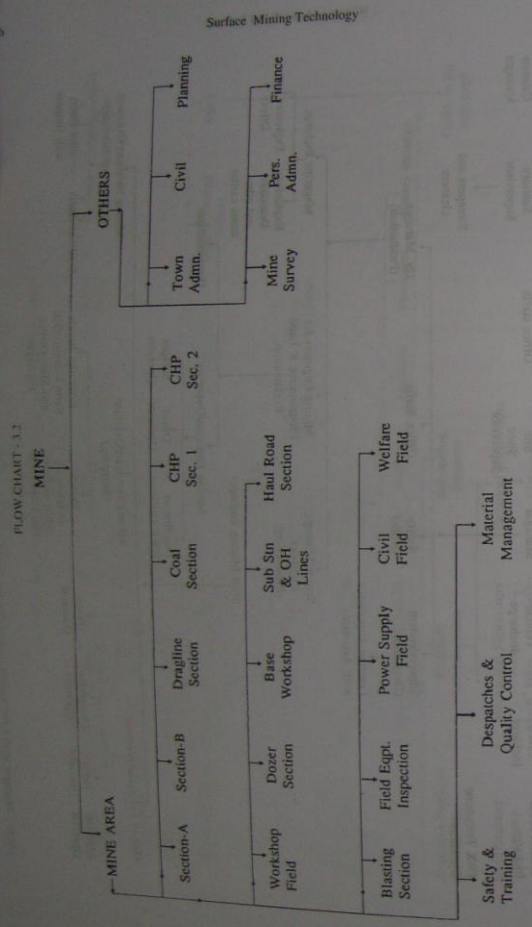
Flow Chart - 3.1 continued

(Note : Manager shall have a good controllable links with the Executive Engineer (Production) and Engineer (Maintenance)).

FLOW CHART - 3.2



CHAPTER 4

VARIOUS LAYOUT PROBLEMS AND THEIR SOLUTIONS

In this chapter various types of surface mining layout problems has been solved based on the knowledge gained from the various chapters which are elaborately discussed in this book. Sixteen layout problems have been selected in the chapter considering most of the aspects of surface mining technology and their parameters. Layouts are drawn in such a manner that even a layman will understand the same. Based on the techno-economics, production and productivity points of view these layouts are the best among the lots. Since various techno-economical, socio economical, geo-technical parameters varies from place to place, country to country, these layouts may be modified a little bit depending upon the site specific conditions. But the basic layout concept will remain more or less the same. Hence these layouts will definitely help as a guideline to design and plan any surface mining layout.

LAYOUT PROBLEM-1

Draw a layout for a surface limestone mine with an output of 10000 Tonne/day in a flat deposit. Assume your own conditions.

SOLUTION

Introduction

Let following points are considered:

- The thickness of limestone deposit 30m
- Gradient :Flat
- Overburden-Top soil with thickness around 3m to 4m
- Deposit is quite sufficient and the life of mine is around 30 years.
- Railway station is very nearby.

The cement plant is located within 2 km distance. It's life is 30 years. If the cement plant is not located very near to the limestone mine, mechanization of mine will not be possible since cost of limestone (2002 price) is only around Rs 109/- per Tonne of the ore (Table No. 3.25)

Description of the Mining Layout

For the production of ore of 10000 Tonne/day (from Table No.3.11) number of production benches/faces required =2.
Height of mineral bench= 10 m
Slope of the mineral bench=65°

Considering the transporting distance around 200m, use dozer cum ripper combination to excavate and transport of loose top soil in the transverse direction (consulting table No. 3.10) in both the sides as shown in Fig. 4.1.

Since deposit is flat, 10000 Tonne of ore production per day at a haul distance of around 2 km is necessary and also since the cost of the limestone ore per tone is only Rs. 109/- rail transport system will be the cheapest transporting mean among all the other modes of transporting system.

From table No. 3.15 ,the best match for production will be shovel and rail transport system. In the table No. 3.17, it is given that the most minimum production per day (with 2 shift operation) by the 3.2 m^3 electro-hydraulic shovel with 35 Tonne dumper will be around 1750 m^3. With rail transport combination and 3 shift operation it will be around 2800 m^3 or more .

From table No. 3.21, it can be seen that the approximate specific gravity of limestone (in situ) is 2.6.

Hence bulk density will be around 2 (say). Therefore, 2 electro-hydraulic shovel of bucket capacity around3.2 m^3 will be sufficient to produce that amount of (10000 Tonne of lime- stone) ore. Total number of shift here considered is 4, out of which there are 3 production shifts and 1 maintenance shift. From the table 3.24 it may be assessed that there

will be not very much cost difference between a shovel of capacity of 3.2 m^3 and 3.6 m^3. Hence purchase 3.6 m^3 shovel which has much higher potential of production and productivity.

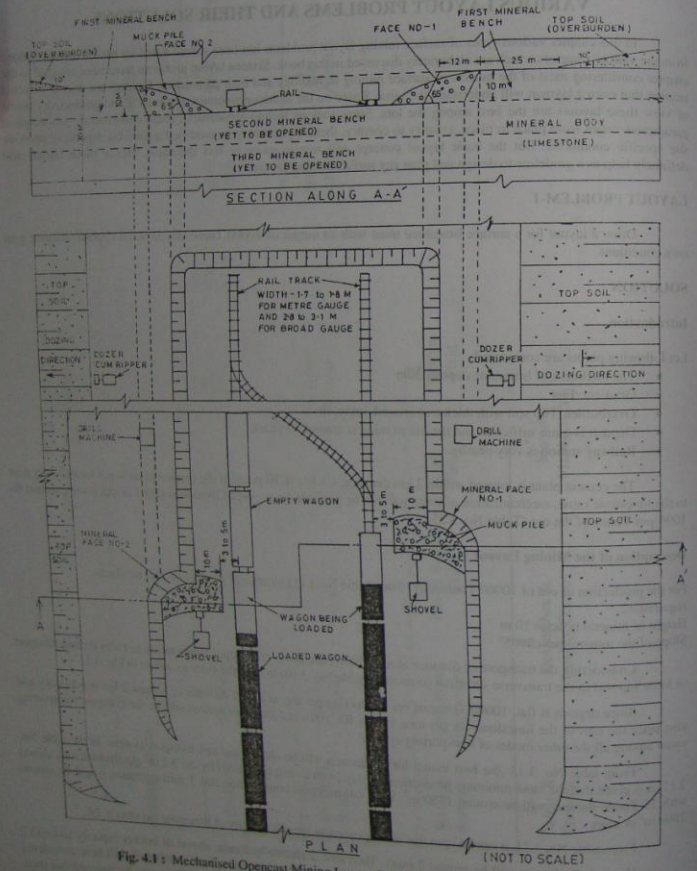


Fig. 4.1 : Mechanised Opencast Mining Layout for High Production (For Flat Deposit)

Consulting table No. 3.28, it is found that the maximum amount of investment for plant and machinery in the first phase (10 years) is Rs 39.24 Crore. Since the life of the mine is 30 years, quite a lump sum money can be spent for the development work, other infra structural facilities and also for purchasing plant and machineries.

Main Equipment

Consulting table No. 3.24, it can be estimated that equipment required to purchase 10000 Tonne of ore will be as follows:

Main equipment	Total numbers	Number x Cost of each item (Rs. in Crore)	Total cost (Rs. in Crore)
3.6m^3 electro hydraulic shovel	2	2x 1.2	2.4
100 mm. dia. Drill machine	2+ 1 (standby) = 3	3x0.26	0.78
25.1 Tonne Roller	1	1x0.26	0.26
Diesel Jeep	2	2x0.05	0.10
Maintenance Van	1	1x0.07	0.07
Fuel truck	1	1x0.05	0.05
Explosive van	1	1x0.07	0.07
35 Tonne Rear Discharge Dumpers for box cut and for material transport	2	2x0.66	1.32
Pump 80 l/s, 112 KW, 550 V	1+1 (standby) = 2	2x0.04	0.08
5 MVA Transforms	2	2x0.18	0.36

Note : 1. Approximate price of the machine enlisted here is based on the price of the financial year 2002. Hence it must be revised every year.
2. Investment criteria is based on the sale value of per Tonne of ore/coal/mineral.

Other Equipment to be Procured

- Dozer cum Ripper combination- 2 Nos.
- Compressors (7 to 9m^3/min capacity and number depends upon the quantity of compressed air requirement)
- Jack hammer drills -3 Nos.
- Water sprinkler -1 Nos.
- Portable workshop containing all the equipment for repairing including portable electric welding set, gas cutting arrangement, etc. .
- Electrical equipment in the substation like insulators, conductors, transformer, oil circuit breaker, carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, etc. Flexible cables for various equipment.
- Exploders, circuit testers, etc.
- Safety appliances and equipment
- Equipment and medical appliances for first aid.
- Office equipment like computers along with its printers , various software and other accessories , Xerox machine, cyclostyle machine, blue printing machine, surveyor's equipment and tools, various monitoring equipment, etc.
- A small workshop along with necessary machines and equipment for repair, maintenance and overhauling of heavy earth moving and other mining machineries.

The mine management has also to come under contract agreement with the railway department for the laying out of the railway lines and also supplying the railways wagons everyday to the mine management. For this reason the management has to pay a quite lump sum amount of money to the railway department.

Method of Working

First of all remove any tree or bushes present over the site of the quarry, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting operation. Scarify the ground. After scarification is over doze the loosen soil/rock with the help of bulldozer. Make clean the mining site. A box cut is to be excavated over the soil and the limestone deposit along the longitudinal direction dividing equally the whole property. Box cut is to be made by drilling and blasting method. Mucking and transporting of both overburden top soil and the limestone will be done with the help of shovel and dumper combination along with the dozer. When sufficient bench width will be available , start winning of limestone mineral and at the same time start excavation of dozing and stacking properly the top soil around the boundary of the deposit.

This dozing is to be done in the transverse direction in both the faces of the bench. A clearance space of around 37m to 40 m from the edge of the bench is necessary. Rail tracks are laid longitudinally as shown in the figure 4.1 leaving a clear space around 3 to 15 m from the edge of the bench. This is done taking due consideration of scattering of blast resulted muck pile so that the later does not effect the rail lines and hinder the movement of railway wagons. Shunting scope is provided to divert empty wagons to other face or to store empty wagons there itself. Position of shovel for loading ore onto wagons, movements of face in the longitudinal direction and movement of bench in the transverse direction has been shown in the figure 4.1. After taking one slice of 12 m width of bench all along the longitudinal length rail lines are to be transversely shifted keeping a minimum distance of 13m to 15 m away from the bench edge. After the finish of first bench, second and third benches are to be worked out successively. The ore to be transported by surface locomotives and wagons which are to be tippled by a tippler inside the cement factory.

Drilling and Blasting

From table No. 3.13
Height of the bench: 10m
Spacing: 4m.
Burden: 3m
Number of rows/blast : 4.
So width of blast =3 x 4 = 12m. No. of faces =2
Drilling and blasting is necessary for one week duration.

So length of blast = $\frac{6x5000}{12x10x2.6}$ = 96.15m ≈96m

Number of holes to be blasted= $\frac{96.15}{4}$ x4 = 96.15 ≈ 96

In a 4 shift working (3 production shift and 1 maintenance shift) a drill machine (DTH) can easily drill 96x10m + sub-grade drill length (0.3X96 = 28.8m) =988.8m total hole length. But one standby drill machine is to be kept in reserve for the purpose of continuity of operation.

Diameter of holes = 100mm
Patterns of blast holes =multiple 'V' cut
Explosive used - Charge per delay - O.C.G.- 1000Kg
Indogel-230 Kg. ANFO-1550 Kg
Initiation - Both down line and trunk links of Cordtex (detonating fuse) and 6 to 10 number delays are used.

Pumping and Drainage System

The pit floor should be sloped (around 2°) towards the extraction side to drain out rain water during the monsoon. Drain and ditches are to be excavated if necessary for flowing of water towards the sump. For proper drainage of the mine water and also to accommodate the rain water, a sufficient capacity sump should be located at the lowest and end point of the pit. Capacity of the sump should be of the 24 hours average rainfall during the monsoon in a year. Let

average total accumulated water per day during the monsoon is around 1500m^3/day. Hence capacity of the sump shall be around 1500m^3. The pump in this case selected is multistage centrifugal pump. If average running time of a pump is around 8 hours then quantity of water to be pumped per hour is 1500/8 = 187.5 m^3/hour =52.08 l/s.
Hence a multistage centrifugal pump having a discharge capacity around 80 l/s, 50m head, 112 kW motor power will be sufficient to handle that amount of water. The pump will have also more potential to handle more quantity of water if there is heavy rainfall. But as a precautionary measure, one standby pump is to be kept in reserve. If necessary one more centrifugal pump of capacity around 2001/s along with a pontoon is to be kept reserve to deal any emergency situation.

In most of the cases, it has been seen that excavation of the mine floor (except a small patch for the purpose of suction range of the pump only) for making sump of 24 hours accumulated water capacity is not possible. Therefore, the lowest point of the mine is slanted in such a manner that water is accumulated at a corner point at the bottom most place from where water is pumped out with the help of pontoon mounted centrifugal pumps. But if it is found that heavy amount of water percolates from the mineral body/overburden ,800 mm to 1200 mm dia. bore holes are to be sunk into the strata which are linked with casings and accumulated water is pumped out from the boreholes with the help of around 400mm to 500 mm dia. borehole pumps.

Land Reclamation and Revegetation

After totally winning the limestone ,the mined out area is to be filled up with the overburden material if it is available from the nearby opencast mine since in this case no overburden except top soil is available in insufficient quantity. If overburden dump (boulders) are not available from the nearby mine , part of the opencast mine is to be filled up with a 0.5m thick filler layer. Over which layers of soil is to be laid leveling up to the surface according to the chronological sequence as it was in pre-mining condition. The dozers are to be fully utilized starting from the dozing, filling, leveling and for making final grading (1 in 20) towards the unfilled area. Before trees plantation bio-fertilization of top soil is very essential. The filled portion of the excavated area is to be ameliorated, bio-fertilized by the agents for generation of micro biota. After bio-reclamation, the reclaimed land shall be developed into a pasture land and a-forestation is to be carried out by the plantation of fast growing tree plants like cassia auriculata, bambusa, nana, acacia nilotica, dalbergia sisoo, prosopis juliflora etc. The unfilled portion of the mined out area should be converted into water pond/lake for water harvesting, fish firming, recreation facilities like swimming pool, scenic spot, etc.

Lighting and Fencing Arrangements

Enough lighting arrangements are to be done both in side the pit and over the surface as per the norms laid down by the DGMS. Similarly enough fencing arrangements are to be done in the outside the boundary of the pit as well as surrounding the strategic places and machineries of the mine as stipulated by the DGMS. Annexure 3.5 and annexure 3.4 may be consulted for these purposes.

Supply of Electricity - Consult annexure 3.3. "A brief note of generalized power supply system in the surface mines".

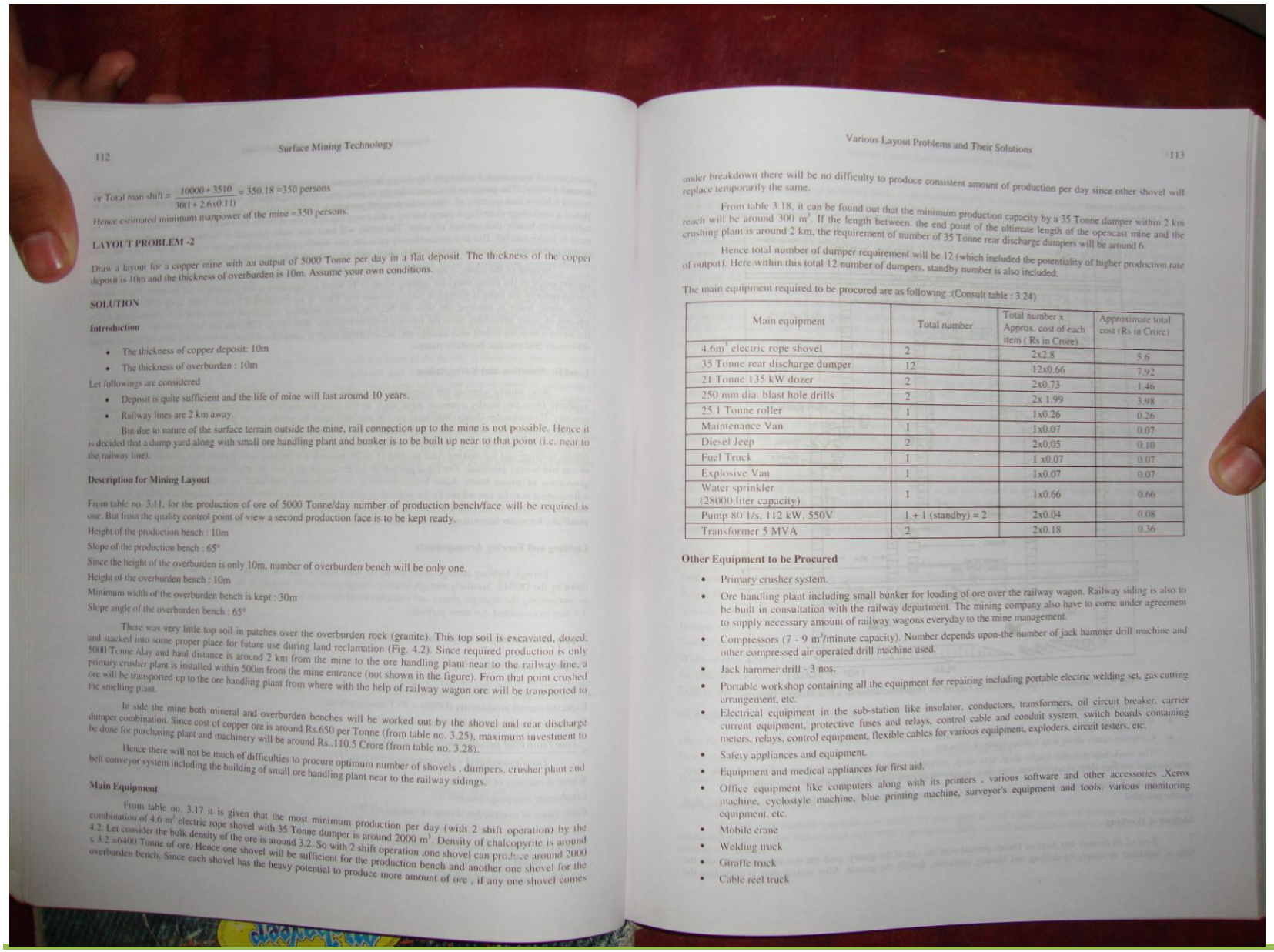
Production and Productivity

Production (given) = 10000 t/day
Expected overall productivity (OMS) = 30 Tonne/man/shift
It is known that overall OMS of opencast mine

$$= \frac{coal/\min eral\ production\ in\ tonne + dx overburden\ in\ m^3}{Total\ manshift(1 + dx stripping\ ratio\ in\ m^3/t)} \quad ...(1)$$

Where, d =density of limestone
Let ultimate stripping ratio will be:
Cubic meter of overburden :Tonne of mineral =0.11:1
Let density of limestone =2.6.t/m^3
Putting all the above values in the equation (1)

$$30 = \frac{10000 + 2.6x1350}{Total\ manshift\ (1 + 2.6x0.11)}$$



or Total man shift = $\frac{10000+3510}{30(1+2.6\times0.11)} \approx 350.18 \approx 350$ persons

Hence estimated minimum manpower of the mine =350 persons.

LAYOUT PROBLEM -2

Draw a layout for a copper mine with an output of 5000 Tonne per day in a flat deposit. The thickness of the copper deposit is 10m and the thickness of overburden is 10m. Assume your own conditions.

SOLUTION

Introduction

- The thickness of copper deposit: 10m
- The thickness of overburden : 10m

Let followings are considered

- Deposit is quite sufficient and the life of mine will last around 10 years.
- Railway lines are 2 km away.

But due to nature of the surface terrain outside the mine, rail connection up to the mine is not possible. Hence it is decided that a dump yard along with small ore handling plant and bunker is to be built up near to that point (i.e. near to the railway line).

Description for Mining Layout

From table no. 3.11, for the production of ore of 5000 Tonne/day number of production bench/face will be required is one. But from the quality control point of view a second production face is to be kept ready.

Height of the production bench : 10m

Slope of the production bench : 65°

Since the height of the overburden is only 10m, number of overburden bench will be only one.

Height of the overburden bench : 10m

Minimum width of the overburden bench is kept : 30m

Slope angle of the overburden bench : 65°

There was very little top soil in patches over the overburden rock (granite). This top soil is excavated, dozed, and stacked into some proper place for future use during land reclamation (Fig. 4.2). Since required production is only 5000 Tonne /day and haul distance is around 2 km from the mine to the ore handling plant near to the railway line, a primary crusher plant is installed within 500m from the mine entrance (not shown in the figure). From that point crushed ore will be transported up to the ore handling plant from where with the help of railway wagon ore will be transported to the smelting plant.

In side the mine both mineral and overburden benches will be worked out by the shovel and rear discharge dumper combination. Since cost of copper ore is around Rs.650 per Tonne (from table no. 3.25), maximum investment to be done for purchasing plant and machinery will be around Rs..110.5 Crore (from table no. 3.28).

Hence there will not be much of difficulties to procure optimum number of shovels , dumpers, crusher plant and belt conveyor system including the building of small ore handling plant near to the railway sidings.

Main Equipment

From table no. 3.17 it is given that the most minimum production per day (with 2 shift operation) by the combination of 4.6 m^3 electric rope shovel with 35 Tonne dumper is around 2000 m^3. Density of chalcopyrite is around 4.2. Let consider the bulk density of the ore is around 3.2. So with 2 shift operation ,one shovel can produce around 2000 x 3.2 =6400 Tonne of ore. Hence one shovel will be sufficient for the production bench and another one shovel for the overburden bench. Since each shovel has the heavy potential to produce more amount of ore , if any one shovel comes

under breakdown there will be no difficulty to produce consistent amount of production per day since other shovel will replace temporarily the same.

From table 3.18, it can be found out that the minimum production capacity by a 35 Tonne dumper within 2 km reach will be around 300 m^3. If the length between, the end point of the ultimate length of the opencast mine and the crushing plant is around 2 km, the requirement of number of 35 Tonne rear discharge dumpers will be around 6.

Hence total number of dumper requirement will be 12 (which included the potentiality of higher production rate of output). Here within this total 12 number of dumpers, standby number is also included.

The main equipment required to be procured are as following :(Consult table : 3.24)

Main equipment	Total number	Total number x Approx. cost of each item (Rs in Crore)	Approximate total cost (Rs in Crore)
4.6m^3 electric rope shovel	2	2x2.8	5.6
35 Tonne rear discharge dumper	12	12x0.66	7.92
21 Tonne 135 kW dozer	2	2x0.73	1.46
250 mm dia. blast hole drills	2	2x 1.99	3.98
25.1 Tonne roller	1	1x0.26	0.26
Maintenance Van	1	1x0.07	0.07
Diesel Jeep	2	2x0.05	0.10
Fuel Truck	1	1 x0.07	0.07
Explosive Van	1	1x0.07	0.07
Water sprinkler (28000 liter capacity)	1	1x0.66	0.66
Pump 80 l/s, 112 kW, 550V	1 + 1 (standby) = 2	2x0.04	0.08
Transformer 5 MVA	2	2x0.18	0.36

Other Equipment to be Procured

- Primary crusher system.
- Ore handling plant including small bunker for loading of ore over the railway wagon. Railway siding is also to be built in consultation with the railway department. The mining company also have to come under agreement to supply necessary amount of railway wagons everyday to the mine management.
- Compressors (7 - 9 m^3/minute capacity). Number depends upon-the number of jack hammer drill machine and other compressed air operated drill machine used.
- Jack hammer drill - 3 nos.
- Portable workshop containing all the equipment for repairing including portable electric welding set, gas cutting arrangement, etc.
- Electrical equipment in the sub-station like insulator, conductors, transformers, oil circuit breaker, carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, flexible cables for various equipment, exploders, circuit testers, etc.
- Safety appliances and equipment.
- Equipment and medical appliances for first aid.
- Office equipment like computers along with its printers , various software and other accessories .Xerox machine, cyclostyle machine, blue printing machine, surveyor's equipment and tools, various monitoring equipment, etc.
- Mobile crane
- Welding truck
- Giraffe truck
- Cable reel truck

- Tyre handler
- Power sweeper

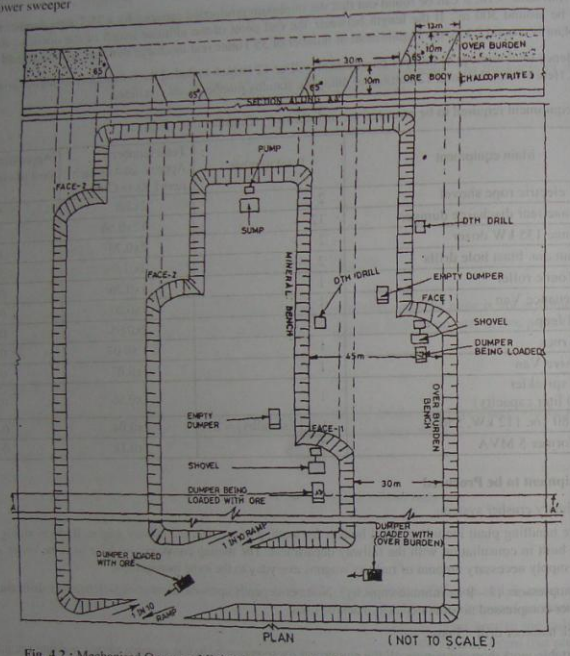


Fig. 4.2 : Mechanised Opencast Mining Layout for High Production in a Flat Mineral Deposit

- Steam cleaning machine
- Wheel motor dolly
- Computer centre along with various kinds of software.

The workshop includes trucks shop, tyre shop, component shop, auxiliary equipment shop, machine shop, structural and welding shop, electrical repairing shop, engine shop, light vehicle shop, carpentry and paint shop.

In the workshop besides various equipment E.O.T. crane, fork lifter, floor mounted hydraulic crane, etc. shall also be provided.

Method of Working

First of all remove any trees or bushes present over the site of the quarry, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting operation. Scarify the ground. After scarification is over doze the

loosen soil/rock with the help of bulldozer. Make clean the mining site. A box cut is to be excavated over the overburden rock (granite) and chalcopyrite mineral body along the longitudinal direction dividing equally the whole property by the drilling and blasting method. Mucking and transporting of both the overburden rock and the ore will be done with the help of shovel and dumper combination. When sufficient bench width will be available, start winning of overburden in full swing first in one direction and make a clearance of nearly 30m to start the operation of the winning of chalcopyrite mineral. For proper utilization of the 4.6 m^3 bucket capacity rope shovel, drilling and blasting operation shall be carried out in such a manner and in such a width that a 15m to 16m width of the face is available for excavation by a shovel. In the figure 4.2 it is clearly shown that a ramp (gradient I in 10) around 6 to 7 time the maximum width of the dumper is excavated to reach the overburden bench and similarly a ramp (gradient I in 10) around 4 to 5 times the maximum width of the dumper is made from the floor of the overburden bench to the floor of the mineral bench. In the first phase after advancing face no. 1 of both the mineral and overburden bench to a considerable distance, start winning of face no. 2 of both the overburden and mineral bench to make them stand-by and for better quality control. Select proper dumping place of the overburden only in one side (right hand side i.e. in the side of the face-1) outside the boundary of the mine. But when sufficient space will be available on the floor of the mineral bench after extraction of minerals , start dumping of the overburden over the floor of the mineral bench starting from the middle point. It will minimize the hauling distance of the overburden as well as will eliminate rehandling of that overburden. Try to finish both the overburden and mineral bench faces (no.1) in one direction only which will systematize the reclamation of overburden rock in the opencast mine. Overburden rocks of face no. 2 can directly be dumped on the floor of the mineral bench.

Ultimate slope of the pit will be around 40° to 45°. Fill up the dematerialized area with the overburden rock stacked outside the boundary of the mine after extraction of the total mineral. Movement of the faces is in the longitudinal direction and the movement of the benches is in the transverse direction. The ore is transported by the dumpers and dump over the primary crushing plant to crush the ore in the size of 150mm. From the crusher plant , ore is transported by the belt conveyor to the material handling plant. From the material handling plant ore is loaded onto the railway wagon through the bunker by apron feeder. Floor of the overburden bench shall be inclined at an inclination of 1 in 20 towards the extraction side. During excavation, the floor of the mineral shall be inclined towards the sump so that water during the monsoon season can directly fall over the sump. If necessary right from the surface up to the sump, drains and ditches are to be formed for free flowing of water.

Drilling and Blasting

Drill machine to be selected here is rotary percussive drills to drill holes of diameter 250 mm. (from table 3.13)

Height of the overburden bench: 10m

Spacing: 6.5m.

Burden: 6m

Number of rows/blast =2

Width of blast =2x6 = 12m

Number of face =1

Drilling and blasting is necessary for one week duration.

So length of blast = $\frac{5000x6}{12x10x4.2} = 59.52m$

(Here the density of overburden rock is also considered around : 4.2 Tonne /m^3)

Hence equal amount of overburden rock and mineral to be excavated per week.

Number of holes to be blasted per week = $\frac{59.52x2}{6.5} = 18.31 \approx 19$

So total length of holes = 19x10+ (sub-grade drilling = 0.3x 19) = 195.7m

A DTH drill machine can drill very easily 1.5 to 2 times of this small amount of drilling length with two production shifts and one maintenance shift. With 3 production shifts and one maintenance shift it can drill even 3 times more of that amount.

Mineral Bench

Height of the mineral bench: 10m

Spacing: 6m, burden: 6m

Number of rows/blast =2

Width of blast =2x6 = 12m

Number of face = 1

Drilling and blasting is necessary for one week duration

So length of blast in one week $= \frac{5000 \times 6}{12 \times 10 \times 4.2} = 59.52\text{m}$

Number of holes to be blasted/week $= \frac{59.52 \times 2}{6} = 19.84 \approx 20$

So total length of hole in the mineral bench in one week

= 20x10 m + (sub-grade drilling =0.3x20) =206m

Therefore it is very clear that one DTH drill machine is sufficient to drill those total length of holes of the overburden and mineral bench. But on stand-by point of view, total number of drills are to be selected 2 (one for overburden bench and other one for mineral bench).

- Pattern of blast holes = multiple 'V' cut
- Explosive used :Indoboost (HE) and Indogel – 270
- Initiation - Both downline and trunk links of Cordtex (detonating fuse) and 6 to 8 number delays are used.

Some Salient Points Regarding Blasting Practices

At present in most of the mine aluminized slurry explosive or site mixed slurry explosive along with booster like O.C.G., toe blast, Bharat toe, etc. are widely in use. The percentage of explosives may be as follows:

40% to 50% OCG and 60% to 50% Site Mixed Slurry Explosive, OCG 40% and PG 2 60%, Toe blast 40% and Supergel 60%, Bharat Toe 40% and Bharat Base 60%, NCN 600 40% and NCN 500 60%, Mex Blast 40% and Mexam 1 & 11 60%, combination of Indo-boost 30%, Indogel 250 40% and Indogel 210 30% etc .Drilling patterns may be in line or staggered and firing pattern may be In-line, square grid In-line, square grid 'V' pattern, staggered grid 'V' pattern, transverse cut pattern, wedge blasting pattern, etc. Delay interval between the rows may be 75 ms.

Mine Drainage and Pumping System

For proper drainage of mine water and also to accommodate rain water during monsoon a sufficient sized sump near to the endpoint of the bench had been sunk. Capacity of the sump should be the 24 hours average rainfall in the monsoon in a year. Let the average accumulated water due to rain fall in a monsoon is around 1500 m^3/day. Hence capacity of the sump shall be 1500 m^3. The pump to be selected is multistage centrifugal pump. If average running time of a pump is around 8 hours, quantity of water to be pumped out per hour is 1500 /8 = 187.5 m^3/hour =52.08 l/s.

Hence a multistage centrifugal pump having a discharge capacity of 80 l/s, 50m head, 112 kW motor power is sufficient to handle those amount of water. It will also have some more potential to handle more amount of water if the rainfall is more. But as a precautionary measure one standby pump is to be kept in reserve. If necessary one big 150 l/s capacity centrifugal pump along with a pontoon are to be procured.

In most of the cases it has been seen that excavation of the mine floor (except a small patch for the purpose of suction range of the pump only) for making sumps of 24 hours accumulated water capacity is not possible. Therefore, the lowest point of the mine is slanted in such a manner that water is accumulated at a corner point at the bottom most place from where water is pumped out with the help of pontoon mounted centrifugal pumps. But if it is found that heavy amount of water percolates from the mineral body/overburden , 800 mm to 1200mm dia bore holes are to be sunk into the strata which are to be lined with casings and accumulated water is pumped out from boreholes with the help of around 400mm to 500 mm dia. borehole pumps.

Land Reclamation and Revegetation

After totally winning the chalcopyrite ore, the void to be filled up by the remaining overburden material which was dumped previously outside the periphery of the mine boundary. The filled overburden is to be dozed by the bulldozer for the purpose of leveling; over which a filler layer of 0.5m thick with ballasts is laid to divert the rain water. Over the filler layer a layer of earth overburden (subsoil) approximate height around 1.5m and at the top of the latter top soil (around 0.5m thick) is to be laid . Layers of soils shall be graded around a gradient 1 in 20. Afterward plants for fast growing trees like cassia auriculata, bambousa nana, acacia nilotica, dalbergia sisoo, prosopis juliflora, etc. are to be planted over the reclaimed land. But before tree plantation amelioration, bio-fertilization of top soil are very essential. Availability of sub-soil, top-soil, etc., it's procurement, transportation and filling are the most important criteria under this heading.

Lighting and Fencing Arrangements

Enough lighting arrangements has to be done in side the pit and over the surface as per the norms laid down by the DGMS. Similarly enough fencing arrangements are to be done in the outside of the boundary of the pit as well as surrounding the strategic places and machineries of the mine as stipulated by the DGMS. Annexure no 3.5 and 3.4 may be consulted in this regards.

Supply of Electricity-Consult Annexure 3.3. 'A Brief Note of Generalized Power Supply System in the Surface Mines'

Production and Productivity - Production (given) =5000 Tonne/day

Expected overall Productivity (OMS) =25 Tonne/man/shift

Estimated minimum manpower requirement will be around 200 persons.

LAYOUT PROBLEM-3

Draw a layout for a surface coal mine with an output of 1000 tonne/day in a deposit having a gradient of 4°. Assume your own condition.

SOLUTION

Introduction

Let following points are considered :

- The thickness of coal deposit: 10m
- Dip of the deposit is 4° and the deposit is outcropped.
- Overburden- Average thickness of top soil is around 3 to 4m.
- Deposit is quite sufficient and the life of the mine is expected around 12 years.
- Railway network is 1.5 km away from the mining site.

Description for Mining Layout

Given that the production of the mine is only 1000 Tonne per day. Cost of coal for that grade (to be produced from the mine) is around Rs 1000/- per Tonne. Hence investment for purchasing plant and machinery which can be done for this mine will be around 24 Crore of rupees (from table 3.28). Since the deposit is outcropped ,gradient is mild and production is not very much, manual method of working will be the best and cheapest proposition.

Since manual method of working is considered, height of the overburden bench (top soil) is the total height of the overburden (here it is 3m to 4m), however, during moving towards the dip side if more height of the overburden is to be negotiated , the number of overburden benches will be more with a height ranging in between 3m to 4m.

Height and slope angle of the coal benches are kept as follows-

Bench	Height	Slop Angle
1st bench	3.0m	50°
2nd bench	3.5m	50°
3rd bench	3.5m	50°

NOTE-(Numbering of benches are started from the bottom), Fig. no. 4.3.

Direct rope haulage will be the cheapest mean of transport system right from the pit to the tippler near to the railway siding located 1.5 km away from the mine.

Since rope haulage system will be used in this case, the width of the each production and overburden benches will be around 10m to 15m.

From Table 3.31 it is very clear that a direct rope haulage having power around 150 kW can very easily produce 1000 m^3 of coal per day. But since a consistent production is desirable and also it is required to handle overburden material which is required to be dumped over the mined out area two haulages of same capacity are to be selected. Total number of shifts may be four (three production shifts and one maintenance shift.)

Main Equipment

The main equipment to be procured

Main equipment	Total number	Number x Approx. cost of each item (Rs in Crore)	Total Approx. cost (Rs in Crore)
Direct rope haulage_150kW	2	2x0.3	0.6
100 mm dia. wagon drills	2	2x0.26	0.52
Dozer (11 tone 67.5 kW)	1	1x0.41	0.41
Diesel Jeep	1	1 x0.05	0.10
Trucks for box cut and also for material transport	2	2x0.07	0.14
Pumps 50 l/s, 35 kW, 550 V	1+1 (Standby) = 2	2x0.02	0.04
500 KVA Transformer (11 KV/550 volts)	2	2x0.2	0.4
1m^3 capacity mine tubs	130	130x0.0015	0.195
1m^3 capacity side tipping trucks	30	30x0.002	0.06

Other Equipment to be Procured

- Compressors-7 to 9m^3/min capacity. Number has to be calculated based on the total amount of compressed air requirement to the wagon drills, jack hammer drills and to other auxiliary systems/machineries.
- Jack hammer drills - 5 Nos.
- A small workshop containing all the equipment for repairing and maintenance. It should also include portable welding set, gas cutting arrangements, etc.
- Electrical equipment in the small sub-station like insulators, conductors, transformer, oil circuit breaker , carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, etc.
- Tippler
- Exploder, circuit testers, etc.
- Equipment for medical appliances for first aid;
- Office equipment like computers along with its printers , various software and other accessories. Xerox machine, cyclostyle machine, blue printing machine, surveyors equipment and tools, etc.

Double tracks for haulage system has to be laid right from the mine upto the tippler. Trucks are also to be laid in all the production and overburden faces and also on the overburden dump which is dumped for land reclamation. Let it may considered that ultimately total meterage of the haulage tracks will be approximately around 7 km. including everything. Let 14 kg rail is to be installed in the mine. Hence cost of 7 km line including installation (from table 3.24) =7 x10 x 0.006 =0.42 Crore rupees.

Method of Working

First of all remove any trees or bushes present over the site of the quarry, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting operation. Scarify the ground. After scarification is over, doze the loosen soil/rock with the help of bulldozer. Make clean the mining site. A box cut is to be made over the little hard top soil and coal outcrop. The box cut is made along the strike direction starting from the outcrop of the coal deposit and to be touched up to the ends of the boundary of the mine. Haulage machine including the haulage building, main haulage tracks, face haulage track, shunting facilities, etc. are to be constructed very carefully. It would be better if main haulage tracks are laid at one end of the mine. Box cut can be made by drilling and blasting operation or simply by manual excavation. After cut making is over (making trenches in the strike-direction), winning operation is to be carried out by simply drilling and blasting operation and loading is to be done manually over the 1m^3 mine tubs. Jack hammer drills can be employed in the first phase of operation like making of box cut and for advancing the face to a considerable distance. One jack hammer can drill on an average 50 number of holes each of 1.5m length. Spacing of holes are 1.2m to 1.5m and burden of holes are 1m to 1.2m. High strength special gelatin explosive of amount 200 to 250 gm/hole is generally used. A powder factor of 10 to 11 Tonne/kg of explosive easily can be obtained. A team of 2 miners are allotted to a space of nearly 2m along whole length of the face in the strike direction to load coal over the 1m^3 mine tub. But at the initial starting point of box making operation, coal is to loaded over the tipping trucks of 8 Tonne capacity.

As the extraction of coal goes on, a time will come when the height of the bench will be more than 3m. At that time a 2nd coal bench will be formed and when this bench will also exceed 3.5m ,a third coal bench will be formed. During the progress of the mining activity overburden top soil will also be touched and an overburden bench has to be formed (Fig 4.3). This overburden top soil will be transported with the help of same haulage system and be dumped over the excavated de-coaled area systematically for the reclamation of land (Fig 4.3). Since there is no tippler over the reclaimed land, side tipping mine tub should be used in this case. All the bench floor shall be inclined at an inclination of 2° towards the excavation for maintaining a good drainage system. The haulage tracks are laid as shown in the fig. 4.3 and the minimum inclination of the main haulage track is maintained at a gradient of 1 in 10. The level haulage tracks near to the faces are kept in a sufficient distance apart to avoid any damage. Proper shunting facilities are provided near to the boundary (outcropped) side of the mine and also if necessary a small haulage may be installed there. Blasting is done in the longitudinal direction in single row in a slice ranging in between 2.5m to 3 m. After taking 1 to 2 slices,the tracks of the mine cars are to be shifted towards the face to minimize the lead distance. The trains of mine tubs (around 20 to 25 in number) are hauled by the direct rope haulages placed at a distance around 1.5 Km away. The loaded train when reaches near to the unloading point, is then shunted and lowered down a little bit and in place of it empty train is connected with the haulage rope and lowered down to the faces of the benches for reloading purpose. The loaded tubs are tippled by the tippler to unload over a bunker from where coal is transported by the railway wagons.

Drilling and Blasting

From table no. 3.13

Height of the bench: 3 to 3.5m

Diameter of holes: 100mm

Burden: 2.5m to 3m

Spacing: 3m to 3.5m

Drill machine used - Wagon drill

Drilling and blasting can be done for one weak production. Let consider the fig. no. 4.3. It has three coal faces and one overburden face. So it is necessary that every production bench must produce a minimum of

1000/3 =333.33 Tonne of coal per day.

So (i) length of blast /bench = $\frac{6x333.33}{3x3x1.3}$ =170.93m

Where density of coal = 1.3t/m^3

Height of the face =3m

Burden =3m

If the height of the coal bench is 3.5m then

(ii) Length of blast/ bench = $\frac{6x333.33}{3.5x3x1.3} = 146.5m$

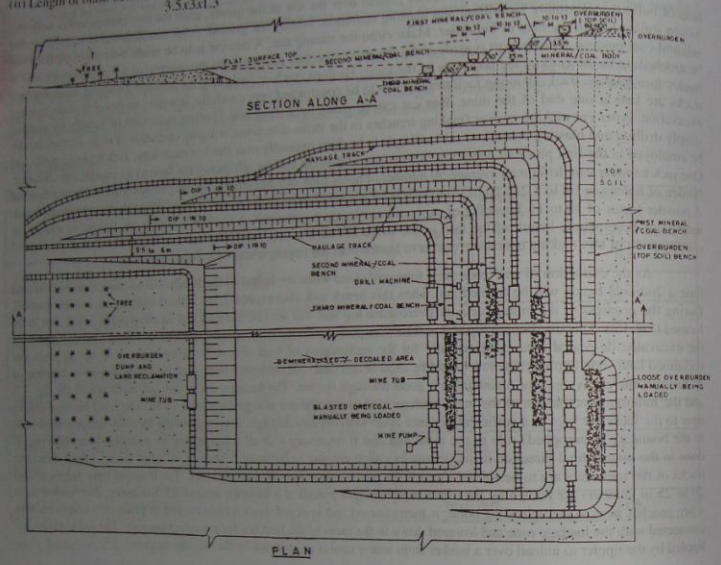


Fig. 4.3: Opencast Mining Layout for Manual Method of working

So total blasting length/week in a coal bench varies in between 146.5m to 170.93m

Number of holes to be drilled/bench=146.5/3=48.8 ≈ 49 holes

to 170.93/3=59.97 ≈ 60 holes per bench .(Here spacing is considered 3m).

Total meterage of holes in each bench varies from 49x3.5 = 171.5m to 60x3 =180m

Therefore, if a wagon drill machine is used for two benches ,one drill machine has to drill only 171.5x2 = 343m to 180x2 = 360m of total hole length. This amount of meterage of hole length can easily be drilled by one wagon drill machines in a week. Hence two wagon drill machines are more than sufficient for this mine. If required a little bit blasting may be necessary for loosening the hard top soil.

Explosive

O.C.G. booster along with Site Mixed Slurry explosives may be very good if used in this mine. However, if procurement of Site Mixed Slurry explosive is not possible ,cartridge type slurry explosive like Energel, Supergel, Mexcol, etc. may be used.

Pumping and Drainage System

The pit floor should be slopped (around 2°) towards the extraction side to drain out rain water during the monsoon. If required drains and ditches are to be excavated for free flowing of water towards the sump. For proper drainage of the mine water and also to accommodate the rain water a sufficient capacity sump is constructed at the lowest but at the end point of pit. Capacity of the sump should be the 24 hours average rainfall amount during the monsoon in a year. Let average total accumulated water per day during the monsoon is around 1000 m^3/day. Hence capacity of the sump shall be around 1000m^3.The pump in this case selected is multistage centrifugal pump. If average running time of the pump is around 8 hours, quantity of water to be pumped per hour is 1000/8 =125 m^3/hour =34.72 l/s.

Hence a multistage centrifugal pump having a discharge capacity around 50 l/s, 50m head and power around 35 kW will be sufficient to deal the said amount of water. The pump will have also more potential to handle more quantity of water if there is a very heavy rain fall. But as a precautionary measure one stand by pump is to be kept in reserve. If necessary another 100 l/s capacity pump is to be procured along with pontoon.

Land Reclamation and Revegetation

It is already mentioned that excavation of coal and soil and also reclamation of decoaled area go hand in hand. But since availability of top soil is not very much, the void can be partially filled. Hence for totally filling the worked out area overburden and soil has to be procured, transported and is to be filled up in the excavated area. But if overburden material from the nearby place is not available , unfilled portion of the mined out area shall be converted into water pond/lake for water harvesting, fish firming, recreation facilities like swimming pool, scenic spot, etc. For fish firming, the water should be properly treated for reducing BOD, COD, N_2, P, grease, toxic elements, etc. with the help of floating hyacinth, fertilization by lime, cow dung, urea, etc are very important. Tilapia mossambica, clarias batrachus, O. striatus, anabas testudineus, catla catla, etc fish species can nicely be cultivated in this water body. The filled up area shall be dozed, leveled and graded (1 in 20) by the dozers. The slope angle of the dumped material may be 37°. The filled portion of the excavated area is to be ameliorated and bio-fertilized by the agents for generation of micro-biota. After bio-reclamation the reclaimed land shall be developed into a pasture land . Afforestation shall be done by the plantation of fast growing tree plants like cassia, auriculata, bambusa nana, acacia nilotica, dalbergia sisoo, prosopis juliflora, etc.

Production and Productivity

Production (given) = 1000 Tonne/day

Expected overall productivity (OMS) = 3 Tonne/man/shift.

$$\text{Overall OMS of opencast mine} = \frac{\text{Coal/mineral production in tonne} + d\times\text{overburden in m}^3}{\text{Total manshift } (1 + d\times\text{stripping ratio in m}^3/\text{t})}$$

Let ultimate stripping ratio (as per Table 3.9) will be (m^3 of overburden : Tonne of coal) = 1.2 : 1

Let density of coal = 1.3t/m^3

$$\text{Hence } 3 = \frac{1000 + 1.3x1200}{\text{Total manshift } (1 + 1.3x\,1.2)}$$

$$\text{Or Total Manshift} = \frac{1000 + 1.3x1200}{3x\,(1 + 1.3x\,1.2)}$$

$$= \frac{2560}{7.68} = 333.33 = 335 \text{ (say)}$$

Hence estimated total manpower of the mine =335

However, if small front end loader of bucket capacity 0.5m^3 of approximate cost around Rs. 0.078 crore or 1.2m^3 bucket capacity front end loaders of costing around Rs. 0.109 crore are used for loading coal and overburden over the mine tub, total manpower requirement will be drastically reduced.

Lighting and Fencing Arrangements

Enough lighting arrangement has to be done both in side the pit and over the surface as per the norms laid down by the DGMS. Similarly enough fencing arrangements has to be done in the outside of the boundary of the pit as well as surrounding the strategic places and machineries of the mine as stipulated by the DGMS. Consult Annexure 3.5 and 3.4.

Supply of Electricity-Consult Annexure- 3.3."A brief note of generalized power supply system in the surface mines".

LAYOUT PROBLEM-4

Draw a layout for an opencast coal mine with an output of 5000 Tonne per day. The thickness and dip of outcropped coal deposit are 20m and 4° respectively. Above the coal seam there is massive sandstone. Also over the sandstone there is the presence of 2m to 3m average thick top soil. Assume your own condition.

SOLUTION

Introduction

- The thickness of coal deposit: 20m
- The dip of the deposit: 4° (1 in 14)
- Overburden is the massive sandstone. Since it is inclined at an inclination of 4°, its thickness will increase according to the depth.
- Over the overburden there is the presence of 2m to 3m average thick top soil.
- Deposit it quite sufficient and the life of the mine will last around 10 years. . Railway links are 2 km away. Due to the nature of the surface terrain rail connection up to the mine is not possible. Hence it is decided that a dump yard along with small coal handling plant with bunker arrangement is to be built up near to the railway line.

Description of the Mining Layout

Let selling price of the coal is Rs 1000/- per Tonne. Hence if mine management wants to produce 5000 Tonne of coal per day, the management can spend around Rs 170 Crore for purchasing plant and machinery (From table no. 3.28)

From Table no. 3.11 for the production of 5000 Tonne of coal/day requirement of number of production benches/faces will be 2.

Height of each production bench (coal) = 10m

Width of the each production bench (minimum) =30m to 35m

Slop of the production bench =65°

Height of the overburden bench = 12m to 15m

Minimum width of the overburden bench =30m to 35m

Slope angle of the overburden bench =65°

Number of overburden benches will be increased as the depth increases (Fig no. 4.4).

Since the mine is very big (medium capacity as per the table 3.8) after scarification is over ,the top soil will be dozed and will be scraped and transported to the proper place and stacked by the scraper.

This top soil will be used during land reclamation. The dumper will transport coal and dump over the hopper of the coal handling plant from where coal will be loaded over the railway wagon via. apron feeder. Both coal and overburden benches will be worked out by the shovel - dumper combination.

Main Machineries

From the table no. 3.17, it is given that the most minimum production per day (with 2 shift operation) by the combination of 4.6m^3 electric rope shovel with 35 Tonne rear discharge dumper is around 2000 m^3. Density of coal is around 1.3t/m^3 whereas the bulk density of coal is around 0.85. With 3 shift operation a 4.6m^3 shovel with 35 Tonne rear

discharge dumper combination will easily produce 3000 m^3 of ROM coal. Hence each shovel will produce 3000x0.85 = 2550 Tonne of coal. Therefore two shovels will produce very easily 2550x2 =5100 Tonne of coal. One more shovel will be required for the overburden bench. Number of shift in the mine is considered is 4 (three production shift and one maintenance shift).

From table no. 3.18, it can be found out that the minimum production capacity by a 35 Tonne rear discharge dumper within 2 Km reach will be around 300 m^3.

Hence requirement of number of 35 Tonne rear discharge dumper for production of coal will be around 600/300 =20. For overburden bench number of dumper requirement will be around 7, So total requirement of dumpers are 20 + 7 + 3 (standby) =30 number

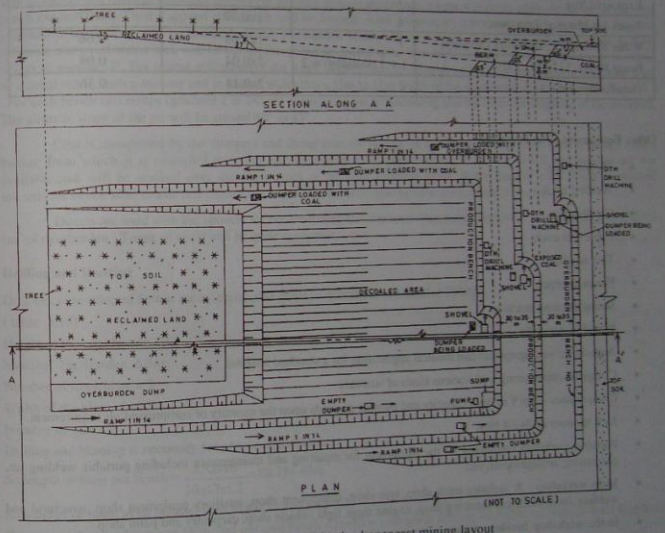


Fig. 4.4 : Highly mechanized opencast mining layout

Main Equipment

The main equipment required to be procured –(Consult-Table 3.24)

Main equipment	Total No.	Number x Approx. cost for each item (Rs. in Crore)	Approximate total cost (Rs. in Crore)
4.6 m^3 Electric Rope Shovel	3	3x2.8	8.4
35 Tonne Rear Discharge Dumper	30	30x0.66	19.8
21 Tonne 135 kW Dozer	2	2x0.73	1.46

250 mm dia. Blast Hole Drills	2	2x1.99	3.98
Motor Grader (109.5 kW)	1	1x0.62	0.62
25.1 Tonne Roller	1	1x0.26	0.26
Scraper (11.5m³ Capacity)	1	1x0.6	0.60
Maintenance Van	1	1x0.07	0.07
Diesel Jeep	2	2x0.05	0.10
Fuel Truck	1	1x0.07	0.07
Explosive Van	1	1x0.07	0.07
Truck for Material Transport	1	1x0.07	0.07
Water Sprinkler (28000 litre capacity)	1	1x0.66	0.66
Pump 80 l/s, 112 kW, 550V	1 + 1 (standby) = 2	2x0.04	0.08
Transformer 5MVA	2	2x0.18	0.36

Other Equipment to be Procured

- Mobile crane
- Welding truck
- Giraffe truck
- Cable reel truck
- Tyre handler
- Power sweeper
- Steam cleaning machine
- Wheel motor dolly
- Ambulance van equipment and medical application for a standard hospital
- Computer centre along with various kinds of software
- Compressors - 7 to 9 m^3/min capacity and number depends upon the quantity of compressed air requirement.
- Jack hammer drills - 4 Nos.
- Portable workshop containing all the equipment for repairing and maintenance including portable welding set, gas cutting arrangements, etc.
- Main workshop - It includes truck shop, tyre shop, component shop, auxiliary equipment shop, structural and welding shop, electrical repairing shop, engine shop, light vehicle shop, carpentry and paint shop
- In the workshop besides various common equipment, E.O.T. crane, fork lifter, floor mounted hydraulic crane, etc. shall be provided.
- Exploders, circuit testers, etc.
- Safety appliances and equipment
- Electrical equipment in the sub-station like insulators, conductors, transformer, oil circuit breaker , carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, etc.
- Flexible cables for various equipment.
- Office equipment like computers along with its printers , various software and other accessories, Xerox machine, cyclostyle machine, blue printing machine, surveyor's equipment and tools, various monitoring equipment, etc.

Method of Working

First of all remove any tree or bushes present over the site of the quarry, grub the tree stumps by the bulldozer or simply by blasting and remove them. Scarify the ground. After scarification is over doze the loosen rock with the help of bulldozer. Make clear the site, expose clear coal and make box cut over the outeropped coal along the strike direction. Box cut is to be done by drilling and blasting method. Excavation of coal is done by the shovel and loaded over the 35 Tonne rear discharge dumpers. Benches are laid along the strike direction and advance along the dip direction. Ramps are excavated in the same gradient (1 in 14) as of the seam in both the ends of the boundary. The width of the roads leading to the benches are 4 times the maximum width of the dumper. To avoid congestion of the traffic, one way traffic system has been introduced in this case. After the box cut is over mucking and transporting of coal will be done with the help of shovel and dumper combination. As the winning process progresses, a time will come when the height of the coal bench exceeds 10m. Then a second coal bench will be formed. After progressing a few meter towards the dip direction overburden stone body will be touched and an overburden bench will be formed. Since there sufficient space will be available ,the overburden rock will be dumped over the de-coaled area. The slope of the dumped overburden in this case is kept to around 37°. For proper utilization of the 4.6m^3 bucket capacity rope shovel drilling and blasting operation shall be carried out in such a manner and in such a width that a 15m to 16m width of the excavating face is available (Fig 4.4) . For each bench two ramps (gradient 1 in 14) are excavated (one for incoming and the other for outgoing of the dumper). The ultimate slope of the pit will be around 40° to 45°.

Coal is transported by the dumpers and dump over the hopper of the coal handling plant. Coal is loaded onto a bunker from which it is unloaded over the railway wagons via. apron feeder. If bunker is full and wagons are not available.coal will be stacked over the dump yard of the coal handling plant. Since floor of each bench is inclined towards the bench side, drains and ditches are excavated for free flowing of water towards the sump.

Dozers are used with the shovels for accumulating the scattered fragmented coal/rock. A stripping ratio of 2.5:1 (m^3 of overburden : Tonne of coal) will be quite economic.

Drilling and Blasting

Drill machine selected here is rotary percussive drills to drill holes of diameter 250mm.

(table :3.13)

Height of the coal bench: 10m

Spacing: 7m. Burden: 6.5m

Number of rows/blast = 2

Width of the blast =2x6.5 = 13m

Number of face =2

Drilling and blasting is necessary for one week duration.

So length of blast per bench = $\frac{2500x6}{10x13x1.3} = 88.75 \approx 89$m.

So number of holes to be blasted/week =(89x2)/7 =25.42 ≈ 26

Hence total length of holes to be drilled per coal bench =26x10 + (sub grade drilling =0.5x26) =273m. So total length of hole in 2 coal benches =273x2 =546m

This meterage of hole length can be drilled very easily by a DTH drill machine in 6 days. As usual the length of blast in the overburden will be also 89m. If the spacing of holes are 7m then number of holes in the bench = (89 x 2)/7 = 25.42 ≈ 26.

Total meterage of holes to be drilled in the overburden bench =26x13 + (sub grade drilling 1.0x26) = 364m. This meterage of hole length can be drilled very easily by a DTH drill machine in one week time.

Pattern of blast holes =multiple 'V' cut.

Explosive used : O.C.G. booster and aluminized slurry explosives

Initiation-Both down-line and trunk-links of Cordtex (Detonating fuse) and 6 to 8 number delays are used.

Mine Drainage and Pumping System

Consult layout problem No.1, 2 and 3

Land Reclamation and Revegetation : Consult layout problem No. 1, 2 and 3.

Production and Productivity

Production (given) =5000 Tonne/day
Expected overall productivity (OMS) = 17.8 Tonne/man/shift
Estimated total manpower requirement =285 person.

Lighting and Fencing Arrangements

Enough lighting arrangements have to be done both in side the pit and over the surface as per the norms laid down by the DGMS. Similarly enough fencing arrangements has to be done in the outside the boundary of the pit as well as surrounding the strategic places and machineries of the mine as stipulated by the DGMS. Also Consult Annexure3.5 and 3.4.

Supply of Electricity-Consult Annexure3.3 "A brief note of generalized power supply system in the surface mines."

LAYOUT PROBLEM -5

A 12m thick manganese ore body is dipping at an inclination of 16° from the horizontal and is outcropped. It is decided to produce 1 million Tonne of ore from the mine per annum. Draw a layout of a mechanized surface mining system. Assume your own conditions.

SOLUTION

Introduction

Let the following points are considered-

- The thickness of rich manganese ore body: 12m
- Dip of the deposit: 16° and the deposit is outcropped
- Overburden rock is quartzites and schists. Over which there is a layer of 1m top soil.
- Deposit is quite sufficient and the life of the mine is expected around 11 to 12 years.
- Railway network is 1.2 Km away from the mining site.

Description for Mining Layout

Given that the production of manganese ore from the mine is 1 million Tonne/year i.e. around 4000 Tonne per day. Cost of manganese ore for that grade (to be produced from the mine) is around Rs 1416/- per Tonne. Hence approximate investment for purchasing plant and machinery which can be done for the mine will be minimum around Rs 169.92 Crore (from Table 3.28).

Since deposit is dipping at 16° and cost of manganese ore is Rs 1416/- per Tonne, a stripping ratio (m^3 of overburden : tonne of mineral) of 3:1 may be expected with high mechanization. Exposing of slices of manganese ore body will be very difficult as the working progresses deeper and deeper. Hence it will always be better if mineral body is exposed off and a 6m bench height is first formed, both for the mineral as well as for the overburden. Later two six meter overburden benches will be clubbed together into one 12m bench for better utilization of 4.6m^3 shovel as well as for using a minimum number of machineries.

In the same way two mineral benches each of 6m height may be clubbed together to form one 12m (maximum) height bench to minimize the use of number of machineries (Fig 4.5). Hence in this case ultimately there will be two overburden benches and one mineral bench. Because the ore body is 16° steep the height of the mineral bench will vary from one end to the other and hence drilling and blasting operation is to be done very carefully and precisely. Although dragline (24m^3 bucket capacity with 96m boom length) would have been the best matched machine in this case for the removal of overburden but the idea can not be implemented since it is a very expensive equipment (present cost is around Rs 95.64 crores). Even cost of dragline of 8-12m^3 bucket capacity and 43m to 61m boom length is Rs 31.81 Crore. Also when production excavator will be under breakdown this dragline will not be a very good proposition to replace the same. Hence it will be better if two overburden benches are worked out with shovel and rear discharge dumper

combination and the mineral bench is worked with front end loader-mobile crusher-mobile belt conveyor-mobile bunker-belt conveyor combination. The properly sized crushed manganese ore (-60mm sizes) will be transported right from the face to the small ore handling plant situated 1.2 Km away near to the railway network. From this place the ore is loaded onto the railway wagons and transported up to the steel industry. Since the transporting system of manganese ore is a series system, if any machine falls under breakdown, the whole system will be stopped and production will be held up. But during that time dumpers from the overburden benches may be diverted for maintaining consistency of production. Hence the system i.e. a mixture of continuous and discrete system will be quite effective.

The height of the overburden bench = 12m
Width of the overburden bench = 35m to 40m
Slope of the overburden bench = 60°
Number of overburden benches = 2
Maximum height of the mineral bench = 12m

Width of the mineral bench = Width of the exposed ore body. Slope of the mineral bench depends upon the maximum height of the mineral bench (12m in this case) and slope of the manganese ore body.

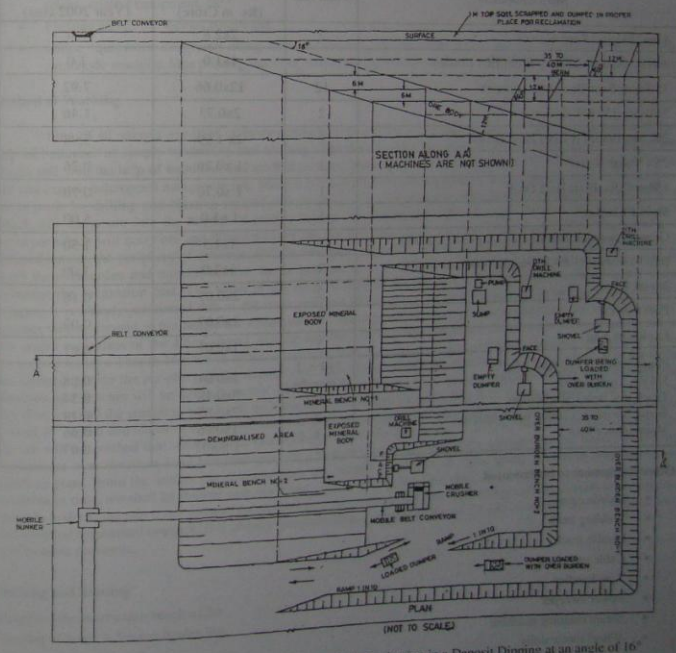


Fig. 4.5 : Mechanised Opencast Mining Layout for High Production in a Deposit Dipping at an angle of 16°

It is given in the table No. 3.17. that the minimum production per day (with 2 shift operation) by the 6.48 m^3 bucket capacity front-end-loader with 35 Tonne rear discharge dumper combination is around 2000 m^3. Since density of manganese ore in-situ is 4.7 tonne/m^3 and the bulk density is around 3.5 tonne/m^3, one front end loader is more than sufficient to produce 4000 Tonne of manganese ore per day. Even lesser capacity front end loader can produce that amount of ore per day. Front end loader should be selected since height of the mineral bench varies sharply. The shovels of overburden benches (12m height) have bucket capacity around 4.6m^3 i.e. a combination of 4.6m^3 electric rope shovel with 35 Tonne rear discharge dumper to be selected in this case. The maximum distance of haul by a dumper to dump the overburden around the periphery of the mine is around 1.5 Km. Hence with each shovel with 6 dumpers will transport and dump a minimum 2100 m^3 of overburden rock.

Therefore, for two shovels number of dumpers required = 2x6 = 12.

Main Equipment

Main equipment required to be procured for the mine are as following :

Main equipment	Total No.	Number x Approx. cost (Rs. in Crore)	Total cost (approx.) (Rs. in Crore) (Year 2002 cost)
4.6m^3 Electric Rope Shovel	2	2x2.8	5.6
6.84m^3 Bucket Capacity Front End Loader	1	1x1.0	1.0
35 Tonne Rear Discharge Dumper	12	12x0.66	7.92
21Tonne 135 KW Dozer	2	2x0.73	1.46
250 mm Dia DTH Blast Hole Drills	2	2x 1.99	3.98
25.1 Tonne Roller	1	1 x0.26	0.26
150mm Dia. Blast Hole Drill	1	1 x0.70	0.70
Mobile Crusher	1	1 x5.0	5.00
Crawler mounted Mobile Belt Conveyor (100m)	1	1x1.5	1.50
Belt Conveyor (1.6 Km length, 1000mm width)	1	1x3.0	3.00
Diesel Jeep	2	2x0.05	0.10
Fuel Truck	1	1x0.07	0.07
Explosive Van	1	1x0.07	0.07
Pumps. 80 l/s, 112 KW, 550 V 5MVA Transformer Truck, for material transport Water Sprinkler (28000 liter capacity) Maintenance Van	1+1 = 2 (1 standby) 3 2 1 1	2x0.04 3x0:18 2x0.07 1x0.66 1 x0.07	0.08 0.54 0.14 0.66 0.07

Other Equipment to be Procured

- Mobile crane ,
- Welding truck ,
- Giraffe truck ,
- Cable reel truck ,
- Tyre handler
- Power sweeper
- Steam cleaning machine
- Wheel motor dolly

- Ambulance van equipment and medical appliances for a standard hospital.
- Computer centre along with various kinds of software
- Compressors-7 to 9 m^3/min capacity and number depends upon the quantity of compressed air requirement
- Jack hammer drills - 4 Nos.
- Portable workshop containing all the equipment for repairing and maintenance including portable electric welding set, gas cutting arrangements, etc.
- Main workshop-It includes truck-shop, tyre shop, component shop, auxiliary equipment shop, machine shop, structural and welding shop, electrical repairing shop, engine shop, light vehicle shop, carpentry and paint shop. In the workshop besides various common equipment, E.O.T. crane, fork lifter, floor mounted hydraulic crane, etc. shall be provided.
- Exploders, circuit testers, etc.
- Safety appliances and equipment.
- Electrical equipment in the substation like insulators, conductors, transformer, oil circuit breaker, carrier current equipment, protective fuses and relays, control cable and conduct system, switch boards containing meters, relays, control equipment, etc.
- Flexible cables for various equipment.
- Office equipment, computers along with its printers , various software and other accessories, Xerox machine, cyclostyle machine, blue printing machine, surveyors equipment and tools, various monitoring equipment, etc.

Method of Working

First of all remove any trees or bushes present over the site of the quarry, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting operation. Scarify the ground. After scarification is over doze the loosen soil with the help of bulldozer and stack it to some place away from the mine boundary. Make clear the mining site and expose outcropped manganese ore. Make a box cut over the outcropped ore body along the strike direction. Box cut is made by drilling and blasting method and excavation is carried out by the shovel and loaded over the dumper which will dump ore into the hopper of mobile bunker. Mobile bunker will discharge ore over the main level belt conveyor which will carry ore and discharge over the ore handling plant. The overburden rock will be transported and dumped outside the periphery of the mine. Slowly overburden benches will be formed and ramps are to be excavated to reach the overburden and mineral benches in the gradient of 1 in 10 only in one end of the boundary of the mine to minimize the excavation cost. The width of the roads leading to the benches are 8 times the maximum width of the dumper. It is already mentioned that as the mine progress in dip direction both the mineral as well as the overburden benches will be formed in 6m height but afterwards two benches will be clubbed together to form 12m height benches. For proper utilization of the 4.6m^3 shovel ,drilling and blasting operation shall be carried out in such a manner and in such a width that a 15m to 16m width of the excavating face is available in the overburden benches. Overburden rocks are dumped outside the periphery of the mining area. Face of the overburden benches will be moved towards the strike direction and benches will be moved towards the dip direction. Whereas for mineral bench faces will move towards the rise direction and the complete bench will be excavated in the strike direction (Fig 4.5). Ultimate slope of the pit will be around 45°. Front end loader of the mineral bench will dump manganese ore into the hopper of the mobile crusher where the ore will be crushed into -60mm sizes and discharged over the mobile belt conveyor. Mobile belt conveyor dumps crushed ore over the main level belt conveyor via a mobile bunker and ultimately unloads into the bunker of the ore handling plant. From the bunker ore is loaded onto the railway wagon via an apron feeder. If railway wagons are not available in time, ore shall be stacked over the dump yard of the ore handling plant. Ditches/drains are excavated on the benches to divert the rain water towards the sump during monsoon. Dozers are used along with the shovels to accumulate the scattered fragmented ore/rock. It is already mentioned that a stripping ratio of 3:1 (m^3 of overburden : Tonne of ore.) will be quite economic.

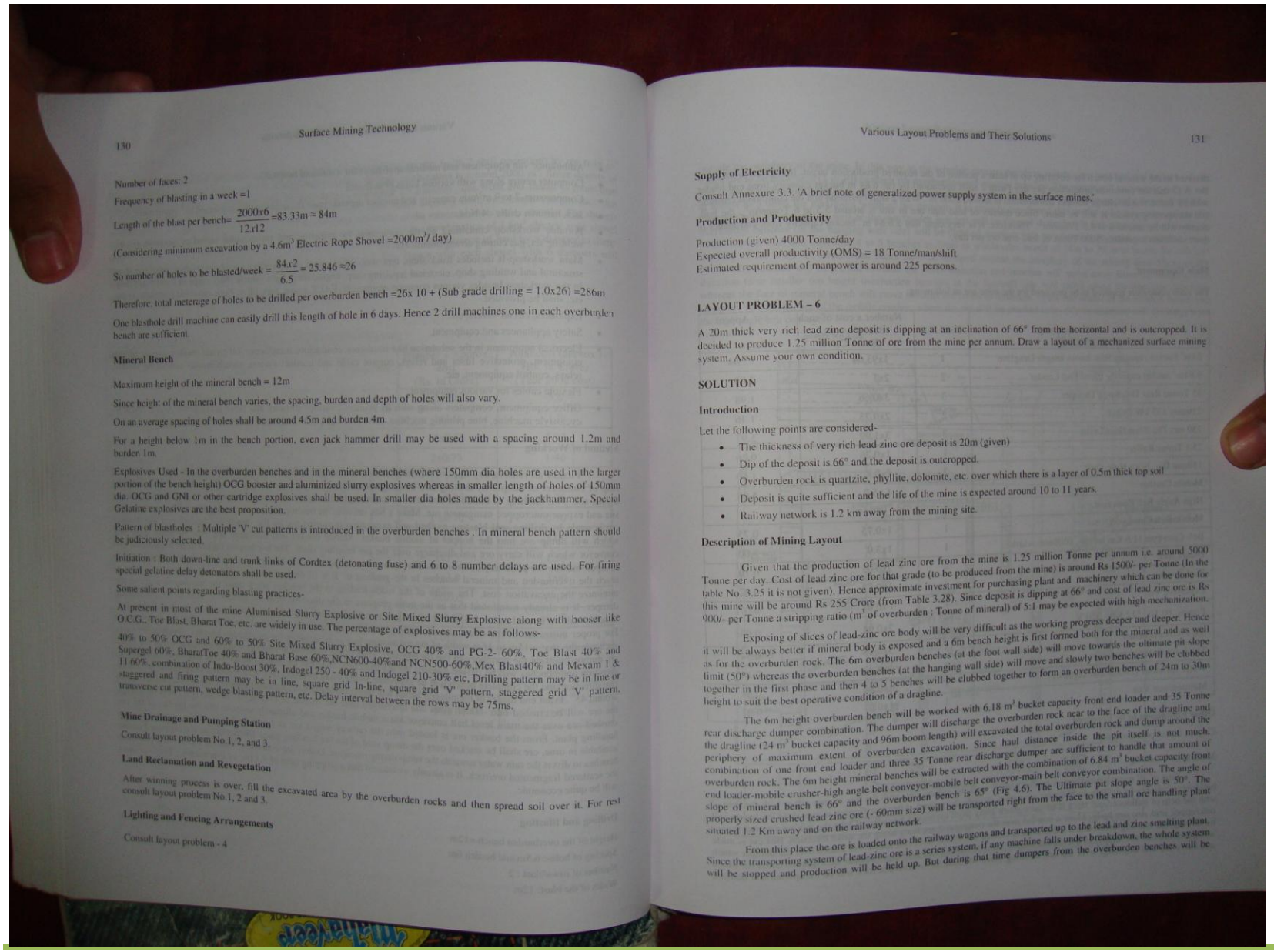
Drilling and Blasting

Height of the overburden bench =12m

Spacing of holes: 6.5m and burden 6m

Number of rows/blast : 2

Width of the blast: 12m



Number of faces: 2

Frequency of blasting in a week =1

Length of the blast per bench= $\frac{2000x6}{12x12}$ =83.33m = 84m

(Considering minimum excavation by a 4.6m^3 Electric Rope Shovel =2000m^3/ day)

So number of holes to be blasted/week = $\frac{84x2}{6.5}$ = 25.846 =26

Therefore, total meterage of holes to be drilled per overburden bench =26x 10 + (Sub grade drilling = 1.0x26) =286m

One blasthole drill machine can easily drill this length of hole in 6 days. Hence 2 drill machines one in each overburden bench are sufficient.

Mineral Bench

Maximum height of the mineral bench = 12m

Since height of the mineral bench varies, the spacing, burden and depth of holes will also vary.

On an average spacing of holes shall be around 4.5m and burden 4m.

For a height below 1m in the bench portion, even jack hammer drill may be used with a spacing around 1.2m and burden 1m.

Explosives Used - In the overburden benches and in the mineral benches (where 150mm dia holes are used in the larger portion of the bench height) OCG booster and aluminized slurry explosives whereas in smaller length of holes of 150mm dia. OCG and GNI or other cartridge explosives shall be used. In smaller dia holes made by the jackhammer, Special Gelatine explosives are the best proposition.

Pattern of blastholes : Multiple 'V' cut patterns is introduced in the overburden benches . In mineral bench pattern should be judiciously selected.

Initiation : Both down-line and trunk links of Cordtex (detonating fuse) and 6 to 8 number delays are used. For firing special gelatine delay detonators shall be used.

Some salient points regarding blasting practices-

At present in most of the mine Aluminised Slurry Explosive or Site Mixed Slurry Explosive along with booser like O.C.G., Toe Blast, Bharat Toe, etc. are widely in use. The percentage of explosives may be as follows-

40% to 50% OCG and 60% to 50% Site Mixed Slurry Explosive, OCG 40% and PG-2- 60%, Toe Blast 40% and Supergel 60%, BharatToe 40% and Bharat Base 60%,NCN600-40%and NCN500-60%,Mex Blast40% and Mexam 1 & 11 60%, combination of Indo-Boost 30%, Indogel 250 - 40% and Indogel 210-30% etc, Drilling pattern may be in line or staggered and firing pattern may be in line, square grid In-line, square grid 'V' pattern, staggered grid 'V' pattern, transverse cut pattern, wedge blasting pattern, etc. Delay interval between the rows may be 75ms.

Mine Drainage and Pumping Station

Consult layout problem No.1, 2, and 3.

Land Reclamation and Revegetation

After winning process is over, fill the excavated area by the overburden rocks and then spread soil over it. For rest consult layout problem No.1, 2 and 3.

Lighting and Fencing Arrangements

Consult layout problem - 4

Supply of Electricity

Consult Annexure 3.3. 'A brief note of generalized power supply system in the surface mines.'

Production and Productivity

Production (given) 4000 Tonne/day
Expected overall productivity (OMS) = 18 Tonne/man/shift
Estimated requirement of manpower is around 225 persons.

LAYOUT PROBLEM – 6

A 20m thick very rich lead zinc deposit is dipping at an inclination of 66° from the horizontal and is outcropped. It is decided to produce 1.25 million Tonne of ore from the mine per annum. Draw a layout of a mechanized surface mining system. Assume your own condition.

SOLUTION

Introduction

Let the following points are considered-

- The thickness of very rich lead zinc ore deposit is 20m (given)
- Dip of the deposit is 66° and the deposit is outcropped.
- Overburden rock is quartzite, phyllite, dolomite, etc. over which there is a layer of 0.5m thick top soil
- Deposit is quite sufficient and the life of the mine is expected around 10 to 11 years.
- Railway network is 1.2 km away from the mining site.

Description of Mining Layout

Given that the production of lead zinc ore from the mine is 1.25 million Tonne per annum i.e. around 5000 Tonne per day. Cost of lead zinc ore for that grade (to be produced from the mine) is around Rs 1500/- per Tonne (In the table No. 3.25 it is not given). Hence approximate investment for purchasing plant and machinery which can be done for this mine will be around Rs 255 Crore (from Table 3.28). Since deposit is dipping at 66° and cost of lead zinc ore is Rs 900/- per Tonne a stripping ratio (m^3 of overburden : Tonne of mineral) of 5:1 may be expected with high mechanization.

Exposing of slices of lead-zinc ore body will be very difficult as the working progress deeper and deeper. Hence it will be always better if mineral body is exposed and a 6m bench height is first formed both for the mineral and as well as for the overburden rock. The 6m overburden benches (at the foot wall side) will move towards the ultimate pit slope limit (50°) whereas the overburden benches (at the hanging wall side) will move and slowly two benches will be clubbed together in the first phase and then 4 to 5 benches will be clubbed together to form an overburden bench of 24m to 30m height to suit the best operative condition of a dragline.

The 6m height overburden bench will be worked with 6.18 m^3 bucket capacity front end loader and 35 Tonne rear discharge dumper combination. The dumper will discharge the overburden rock near to the face of the dragline and the dragline (24 m^3 bucket capacity and 96m boom length) will excavated the total overburden rock and dump around the periphery of maximum extent of overburden excavation. Since haul distance inside the pit itself is not much, combination of one front end loader and three 35 Tonne rear discharge dumper are sufficient to handle that amount of overburden rock. The 6m height mineral benches will be extracted with the combination of 6.84 m^3 bucket capacity front end loader-mobile crusher-high angle belt conveyor-mobile belt conveyor-main belt conveyor combination. The angle of slope of mineral bench is 66° and the overburden bench is 65° (Fig 4.6). The Ultimate pit slope angle is 50°. The properly sized crushed lead zinc ore (- 60mm size) will be transported right from the face to the small ore handling plant situated 1.2 Km away and on the railway network.

From this place the ore is loaded onto the railway wagons and transported up to the lead and zinc smelting plant. Since the transporting system of lead-zinc ore is a series system, if any machine falls under breakdown, the whole system will be stopped and production will be held up. But during that time dumpers from the overburden benches will be

diverted to the mineral bench for covering up at least a portion of the required production target. It is given in the table No. 3.17 that the minimum production per day (with 2 shift operation) by the 6.84 m^3bucket capacity front end loader with 35 Tonne rear discharge dumper combination is around 2000 m^3. With 2 shift operation and the continuous crushing and transporting system it will be more. Since density of lead zinc ore in situ is around 5.5 to 6 Tonne/m^3 , the bulk density will be around 4 to 4.5 Tonne/m^3 . Therefore, it is very clear that a 6.84 m^3 bucket capacity front-end-loader is quite sufficient to produce 5000 Tonne of lead-zinc ore per day.

Main Equipment

The main equipment required to be procured for the mine are as following:

Main equipment	Total No.	Number x cost of each item (Approx.) (Rs. in Crore)	Approx. total cost (Rs. in Crore)
24m^3 bucket capacity 96m boom length Dragline	1	1x93.64	93.64
6.84m^3 bucket capacity Front End Loader	2	2x1	2.00
35 Tonne Rear Discharge Dumper	3	3x0.66	1.98
21tonne 135 kW Dozer	2	2x0.73	1.46
250 mm Dia Blast Hole Drills	3	3x1.99	5.97
25.1 Tonne Roller	1	1x0.26	0.26
150mm Dia. Blast Hole Drill	2	2x0.8	1.60
Mobile Crusher	1	1 x5.0	5.00
High Angle Belt Conveyor	1		
Mobile Belt Conveyor(50m)	1	1x0.75	0.75
Belt Conveyor (1.6 Km length, 1000mm width)	1	1x3.0	3.00
Diesel Jeep	2	2x0.05	0.10
Fuel Truck	1	1x0.07	0.07
Explosive Van	1	1 x0.07	0.07
Pumps, 80 l/s, 112 KW, 550 V	1+1=2 (1 standby)	2x0.04	0.08
5MVA Transformer	3	3x0.18	0.54
Truck for material transport	2	2x0.07	0.14
Water sprinkler (28000 liter capacity)	1	1x0.66	0.66
Maintenance Van	1	1x0.07	0.07

Other Equipment to be Procured

Consult layout problem No.5

Method of Working

First of all remove any trees or bushes present over the mining site, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting operation. Scarify the ground. After scarification is over doze the loosen soil with the help of bulldozer and stack it to some place away from the mine's boundary. Clean the mining site and expose outcropped lead zinc ore body. Make a box cut over the outcropped ore body along the strike direction. Box cut is made by drilling and blasting method and excavation is carried out by the front end loader and loaded over the dumper which will dump ore into the hopper of the mobile bunker. Mobile bunker will discharge ore over the main belt conveyor which will carry ore and discharge over the ore handling plant whereas the overburden rock will be transported and dumped

outside the periphery of the mine. In this way as the winning process progress ,overburden benches will be formed and ramps are to be excavated (1 in 10) to reach the overburden and mineral benches. The ramps will be situated only in one end of the boundary of the mine to minimize the excavation cost. The width of the roads leading to the benches shall be. at least 4 to 5 times the maximum width of the dumper. It is already mentioned that as the mine progresses in the dip direction both the mineral as well as the overburden benches will be formed in 6m height but afterwards the 6m height overburden benches (in the hanging wall side) will be clubbed together with two benches to make a combined bench height around 12m and thereafter 4 to 5 benches to form an overburden bench of 24m to 30 m in height for efficient working of the dragline. Overburden rocks shall be dumped outside the periphery of the mining area. Faces of the overburden benches will be moved towards the strike direction and benches will move away from the hanging wall direction (also smaller 6m height overburden benches at the footwall side will move along the opposite direction) whereas the face of mineral bench will move towards the footwall direction. Front end loader of the mineral bench will dump ore into the hopper of the mobile crusher, where the ore will be crushed into - 60mm sizes and will discharge over the high angle belt conveyor.

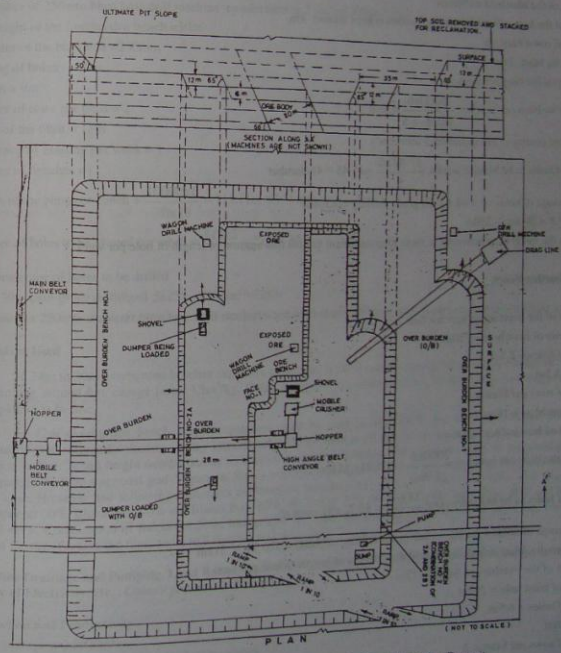


Fig. 4.6 : Mechanised Opencast Mining Layout for Steeply Dipping Deposit

High angle belt conveyors will discharge ore into the mobile belt conveyor. From mobile belt conveyor ore will be discharged into the main belt conveyor via a mobile hopper (or bunker) and ultimately it will be unloaded into the bunker of the ore handling plant (Fig 4.6). From the bunker ore is loaded onto the railway wagons via. an apron feeder. If railway wagons are not available in time ,the ore shall be stacked over the dump yard of the ore handling plant. Ditches and drains are excavated on the benches to divert rain water towards the sump during monsoon.

Drilling and Blasting

(A) Mineral Benches

Height of the mineral bench =6m
Diameter of the blasthole =150mm
Spacing of the hole is 5m whereas the burden is kept around 4m.
Number of rows/ blast : 6
Width of the blast: 24m
Frequency of blasting per week =1

Length of the blast per bench = $\frac{5000x6}{6x24x5.5}$ =38m

(Considered density of ore in situ =5.5 tonne/m^3)

Number of holes to be blasted/week= $\frac{37.87x6}{5}$ =45.45 ≈ 46 number

Total meterage of holes =6 x 46 + (sub grade drilling =0.3x46)
=276 + 13.8 =289.8m ≈ 290m
A 150mm dia. DTH blast hole drill is more than sufficient to drill this amount of length of hole per week.

(B) Overburden Bench

(i.) Height of the overburden bench = 6m
Diameter of blast hole = 150mm
Spacing of holes = 5.5m,
Burden = 4.5m
Number of rows per blast = 4
Width of the blast = 18m
Frequency of blasting per week = 1

Length of the blast per bench = $\frac{2000x6}{6x8}$ = 111.11≈ 111m

Number of holes to be blasted/week= $\frac{111x4}{5.5}$ = 80.72 ≈81 number

Total meterage of holes = 6 x 81 + (Sub grade drilling =0.3x81) = 510.3m ≈ 511m
One 150mm dia. blast hole drill can drill this amount of length of hole per week.
(ii.) Height of the overburden bench = 12m
Diameter of blast hole = 250m
Spacing of holes = 6.5m,
Burden = 6m
Number of rows per blast = 2

Width of the blast = 12m
Frequency of blasting per week = 1
Number of benches =2

Length of the blast per bench= $\frac{5000x6}{12x12}$ =208.33m ≈ 209m

(Maximum excavation by a 24m^3 dragline = 10000m^3)

Number of holes to be blasted/week = $\frac{209x2}{6.5}$ =64.3 ≈ 65 number

Total meterage of holes to be drilled = 12x65 + (sub grade drilling = 1x65) =845m
For two benches 2x845 = 1690m
So number of 250mm blast hole drill machine requirement is 3 (as per Table 3.14).
(iii) Height of the overburden bench =30m
Diameter of the blasthole =250mm
Spacing of holes -= 10m,
Burden = 9m
Number of rows per blast = 2
Width of the blast = 18m
Frequency of blasting per week = 1
Number of benches =1

Length of the blast per bench = $\frac{10000x6}{30x18}$ = 111.11m ≈ 111 m

Number of holes to be blasted per week = $\frac{111x2}{10}$ =22.2 ≈ 23

Total meterage of holes to be drilled
=23 x 30 + (sub-grade drilling=1.5x23) = 724.5m ≈725m
So 3 number 250mm diameter blast hole drill machines are more than sufficient to drill this amount of blast hole length.

Explosives Used

In 24m to 30m overburden benches explosive to be used is OCG and Aluminized Bulk Slurry Explosive. Height of stemming around 6m, charge factor 1.5m^3/kg. Delay interval between rows 15ms. Drilling pattern is staggered and firing pattern is In-line.

In 12m overburden bench height explosives shall be used is 40% OCG and 60% Site Mixed Slurry of Aluminized Bulk Slurry Explosives. Delay interval between the rows is 15 ms. Drilling pattern is staggered and firing pattern is In-line. In 6m height overburden as well as in the mineral benches drilling pattern shall be staggered and the firing pattern shall be staggered grid 'V' pattern. Slurry explosives along with booster explosives may be effectively used in this case. However, the following explosives combination may be used - OCG 40% and PG-2 60%, Toe Blast 40% and Supergel 60%, Bharat Toe 40% and Bharat Base 60%, NCN 600 40%, NCN 500 60%, Max Blast 40% and Mexam 1 & 11-60%, Combination of Indoboost 30%, Indogel250 40% and Indogel 210 30%, etc. Both down line and trunk links of Cordtex (detonating fuse) and 6 to 8 number delays are used.

For Mine Drainage and Pumping, Land Reclamation and Revegetation, Lighting and Fencing Arrangements, Supply of Electricity, etc. : Consult previous layout problems.

Production and Productivity

Production (given) 5000t/day.
Expected overall productivity (OMS) = 15 t /man/shift.
Estimated requirement of manpower is around 350 persons.

LAYOUT PROBLEM – 7

An iron ore deposit is 3 km in length, 1 km in width, 20m in thick occurred at 300m above the ground level in the form of a hillock. The deposit is flat over which there are 8m and 2m overburden consisting of laterite and top soil respectively. The mine is required to produce 2 million Tonne of iron ore. Give a suitable layout of a surface mine. Assume your own condition. .

SOLUTION

Introduction

It is given that

Length of the deposit =3Km

Width of the deposit =1 Km

Thickness of the deposit =20m

Gradient of the deposit =Flat

Deposit occurred at a height of 300m above the ground level in the form of a hillock.

Thickness of overburden = 10m (8m thick laterite and 2m thick top soil)

Density of iron ore (hematite). = 5 tonne/m^3

Total reserve of the mine = $\frac{3000 \times 1000 \times 20 \times 5}{1000000}$ = 300 million Tonne

Since production of iron ore is only 2 million Tonne/annum, the life of the mine will be around 300/2= 150 years.

Railway network which was situated 2 Km away from the hill has been linked up to the floor of the hill and railway siding is constructed under the bunker for directly loading over the railway wagon via an apron feeder.

Description of Mining Layout

It is given that the production of the iron ore from the mine is 2 million Tonne per annum i.e., 8000 Tonne.

per day. The grade of iron ore is very good and let cost of iron for that grade is around Rs.252.00 per Tonne. Since life of the mine is around 150 years a huge amount of money can be invested for the development purpose and also

for the establishment of infrastructural facilities. Stripping ratio i.e. Cubic metre of overburden : Tonne of mineral = 0.1 :1.

Since deposit is situated over the hillock at a 300m above the ground level, if dumpers are selected as a transporting media right from the face to the crushing plant of the ore handling plant near to the railway siding situated at a far off distance, a heavy amount of money is to be spent only for making long and wide zigzag road in the hillock. Due to long distance of travel, negotiating heavy gradient and productivity will be very low and also it will be very unsafe operation.

Hence it is decided that an ore pass will be sunk right from the hill top to the loading point at the foot of the hill. The diameter of the ore pass is around 10m which first sunk vertically and then at an inclination of 55° near to the foot of the hill (Fig 4.7 (B)). Inclination of ore pass (which will act as a bunker too) is made to check the momentum of the crushed ore. At the bottom of the ore pass, steel hopperr with chute arrangement is constructed and at it's bottom an apron feeder is provided for wagon loading .This loading point is constructed by excavating the foot of the hill. Railway siding is also constructed nicely by it's side for railway wagon loading. Crusher plant is located a few meter below from the mouth of the ore pass. The ore pass is situated away from the mining site and in such a position that the maximum travel distance by a dumper is around 3.5 km at the time when the face will reach in the boundary of the strike direction i.e., at about 3 km away from the initial starting point. The ore pass will act as a bunker. Since the railway facility is available near to the mining site, it will be one of the best and cheapest mode of transportation system. But if the mine would have been situated in the remote area where there is no railway link facilities, surface terrain is hilly and if the sea port is 16Km to 20 Km away from the mining site ,for producing 8000

Tonne of ore per day bi-cable ropeway system with an average hourly capacity of 600 Tonne per hour will be a very good solution (Fig 4.8). The initial investment for the bi- cable ropeway system is very high but it's operating cost is

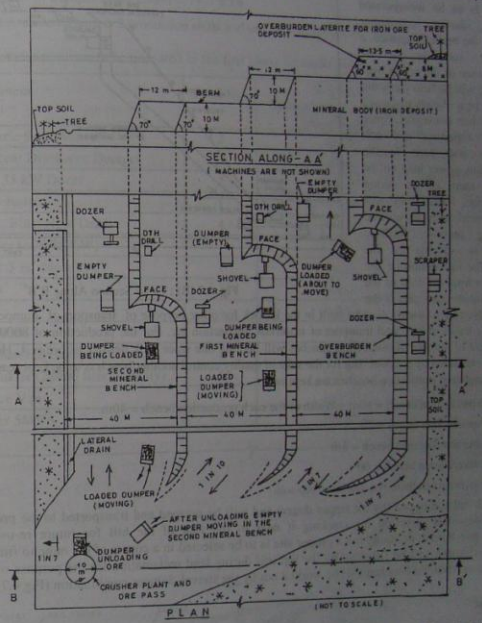


Fig. 4.7 (A) : A Generalised Layout of a Highly Mechanised Surface Mine for a Hill Top Massive Flat Deposit for High Producion (Iron Deposit)

the cheapest among all the transporting systems. Sizes of the primary crushed ore should be small (-150 mm size) enough so that they can well accommodate in the carrier. In the same manner the ore pass near to the top of the hill will be first vertical and then inclined to around 55° to check the momentum of the crushed ore falling from the primary crusher situated a few meter below from the mouth of the vertical ore pass. The other end of the ore pass will be connected with an automatically operated bunker cum hopper arrangement to automatically load the crushed ore into the ropeway carrier. The ropeway will transport ore from the mine to the sea port and unload ore over a bunker (Fig 4.8). Although the service life of a bi-cable ropeway is very high, the main difficulties with this system is that cost of initial capital investment which is very high and it can not transport very high amount of material per hour.

In case both the facilities as mentioned earlier is not possible to implement , another alternative will be left to transport ore by belt conveyor system. Since angle of repose of material is roughly around 17°, the conveyor belt should be arranged to made an inclination less than that amount of gradient. This belt conveyor transport system can be utilized to transport ore at a very high rate as compared to an aerial ropeways but it's installation cost is also very high. Beside the

installation cost another main problem associated with it is that very efficient braking system is to be incorporated with it. But during continuous movement when the brake is applied, a heavy material suddenly to be accumulated in some of the points over the belt and there will be heavy amount of spillage of ore (it is already discussed in the portion of belt conveyor). Since both the aerial ropeways and belt conveyor moves with load in the downward direction, power may not be required to run them except during initial starting time. Rather they will generate power which may be utilized for the lighting purpose. Railway network facility is available for this particular mine .The first option i.e. the rail transport right from the foot of the hill to the steel industry/sea port is the best proposition. Narrow roadways is to be built in the hillock for other purpose of transport of dumpers, shovels and other machineries, travel of men and transport of materials, inspection etc. For the production of 8000 Tonne of iron ore/day requirement of number of production benches will be only one since bulk density of ore is 4. Hence only 2000 m^3 of ore is to be excavated by a shovel. But for the sake of quality control another bench is to be kept ready. Therefore it will be better if number of production benches are kept two.

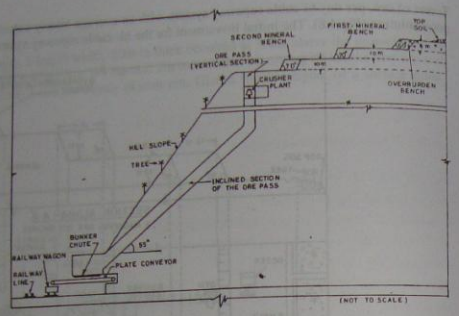


Fig. 4.7 (B) : Cross Section Along B B'

Height of the each production bench = 10m. , Width of the each production bench = 40m.

Slope angle of the production bench = 70°.

Height of the overburden (laterite) bench = 8m

Slope angle of the overburden bench = 60°.

Above the laterite overburden there is a 2m loose top soil.

Top soil will be dozed and ripped (by the ripper dozer) and will be scraped and transported to the proper place and dumped in the downhill direction for stacking it near to the foot of the hill for future re-use during land reclamation. For overburden dumping too a proper site is to be selected in a corner of the mine to fill up the valley. This dumped overburden will be filled up by the top soil during land reclamation for the purpose of revegetation. Both mineral and overburden benches will be worked out by the shovel-dumper combination (Fig 4.7 (A)).

Main Equipment

From table No. 3.17 it can be seen that the minimum production per day (with 2 shift operation) by the combination of 4.6 m^3 electric rope shovel with 35 Tonne rear discharge dumper is around 2000 m^3. Density of iron ore is around 5 (in-situ) whereas the bulk density of iron ore is around 4. So it is very clear that only one face is sufficient to produce 8000 t of iron ore. But from the quality point of view two faces are to be kept ready. So it will be better in the first phase of operation two 4.6 m^3 shovels are procured, one for the production bench and the other one for the overburden bench. Whenever necessary, any one shovel can be shifted to the other production face for quality control since the equipment are new but later on another one shovel to be purchased for the maintenance of consistency of production.

If average transporting distance of overburden removal is 2 Km, a 35 Tonne rear discharge dumper will transport and dump a minimum of 300 m^3 of overburden. Hence with a 4.6 m^3 shovel, for excavating 2000 m^3 of overburden, the number of requirement of dumper will be 2000+300 = 6.66 ≈ 7 number.

Similarly for 2000 m^3 ore (2000x4 = 8000 Tonne) too, if the ultimate length of haul is 3.5 Km. one 35 Tonne rear discharge dumper for this length of haul can produce a minimum of 220 m^3 of ore. Hence number of dumper requirement for this amount of production = 2000/220 = 9.09 = 9 .

This will cover standby facilities too since pit works for a limited period only in the extreme end of the bench (here it is 3 Km).

So number of dumper requirement = 7+ 9 = 16

Number of shift considered here is 2 production shifts and one maintenance shift.

The main equipment required to be procured in the first phase (consult table no. 3.24)

Main equipment	Total No.	Number x Approximate cost of each (Rs in Crore)	Approximate total cost (Rs in Crore)
4.6m^3 Electric Rope Shovel	2	2x2.8	5.60
35 Tonne Rear Discharge Dumper	16	16x0.66	10.56
21 Tonne 135 kW Dozer	2	2x0.73	1.46
25.1 Tonne Roller	1	1x0.26	0.26
150mm dia. Blast Hole Drills	4	4x0.8	3.20
Motor grader (108.75 KW)	1	1 x0.62	0.62
Scraper (11.5 m^3 capacity)	1	1x0.6	0.60
Diesel Jeep	3	3x0.05	0.15
Fuel Truck	1	1x0.07	0.07
Explosive Van	1	1x0.07	0.07
Truck for material transport	1	1x0.07	0.07
Water Sprinkler (28000 liter capacity)	1	1x0.66	0.66
Maintenance Van	1	1x0.07	0.07
Transformer 5MVA	2	2x0.18	0.36

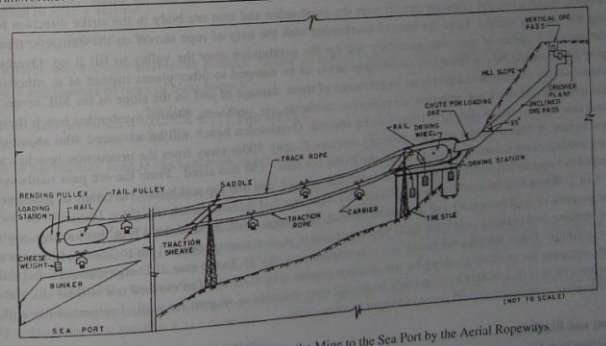


Fig. 4.8 : Transport of Ore Right from the Mine to the Sea Port by the Aerial Ropeways

Beside these a primary gyratory crusher of size receiving opening 105cm x 382.5 cm, capacity 700 Tonne/hour is to be installed to crush the ROM iron ore into - 150 mm size. Loading with chute along with incorporation of feeder arrangement is to be done at the lower most point of the ore pass. Mine management with consultation with the railway department should construct railway siding facilities near to the loading point. Automatic loading arrangements in the railway wagon may also be introduced.

Other Equipment to be Procured

Consult layout problem no. 5

Other Arrangements to be Done in the Mine

Establishment of mine workshop, power sub-station, administrative building, attendance room, rest shelter, mine officers', supervisors' and workers' colony along with a small township, arrangement of water storage, location and laying out of pipe lines, pumps, water treatment plant for drinking water, making of roadways in the colony, offices and other places including connection to the nearby township, location and arrangement of explosive storage magazine, central and site stores, laying of electrical power supply lines including their protection and safety, etc. are to be planned and executed very carefully. Establishment of survey section, material management wing, industrial engineering department, civil section, planning cell, geological department, health centre, training centre, account section, welfare and personnel department, administrative department, security wing, workshop administration, etc. staff management functional wings are not less important and hence sufficient infrastructural facilities in the form of building and other necessary facilities are very essential. In the line management wings (those who are directly involved for production) various wings/sections controlled by the mining and engineering departments are the backbone of the mine. Hence for them sufficient amount of civil construction work and procurement of various amenities are essential for better production and productivity. Therefore it is very clear that a substantial amount of money is to be spent for those purposes.

Method of Working

First of all remove any trees or bushes present over the mining site, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting operation. Scarify and rip the ground with the help of ripper dozer combination. Doze soil, scrap the top soil with the help of scraper, load them in it, transport and unload the top soil from the top of the hillock over a stacking place at the foot of the hill for future reuse during land reclamation. Make a box cut (making trench) over the overburden and iron ore body in the strike direction by the drilling and blasting method. Load the blasted overburden with the help of rope shovel on the dumpers, transport the dumpers in a corner of the mine boundary and tip the overburden over the valley to fill it up. Overburden dumping place should be pre-fixed and overburden never to be dumped to other places instead of it, otherwise it will create severe environmental problems like damage of trees, damage of soil on the slope of the hill, severe land erosion and land sliding, water pollution, air pollution, aesthetic, etc. problems. Slowly overburden bench (height = 8m) and one mineral bench (height = 10m) will be formed. Overburden bench will be advanced 40m ahead of the mineral bench. A 10m diameter ore pass (already discussed) sunk 500m away from the production benches and a primary crusher plant is installed in it to crush the ROM ore into - 150 mm sized . From the ore pass roadways are connected to two mineral benches, one laterite overburden bench and one top soil bench via ramps of gradient 1 in 10 to 1 in 7. When first mineral bench is advanced by 40m ,the 2nd mineral bench (height = 10m) will be formed for the purpose of quality control and as a stand by face. For proper utilization of the 4.6 m^3 bucket capacity rope shovel ,drilling and blasting operation shall be carried out in such a manner that a 15m to 16m width excavating face is available. From the mineral bench ore is transported by the 35 Tonne rear discharge dumpers and unload over the ore-pass for primary crushing by the primary crusher (Fig 4.7 (A)). The crushed ore will be stocked into the ore pass (it acts as a bunker too) and will be loaded over the railway wagons manually / automatically with the help of loading system (chute and apron conveyor combination).

Drilling and Blasting

Production bench

Production per day = 8000 Tonne

Height of the production bench = 10m

Diameter of the blasthole = 150 mm

Inclination of hole = vertical

Burden = 4m

Spacing = 4.5m

Number of production benches = 1 + 1 (one for stand by and quality control).

Number of rows of blast hole = 3

So width of the blasting zone in the bench = 3x4 = 12m

Frequency of blasting per week = 1

Length of blast per bench = $\frac{8000x6}{10x12x5} = 80m$

[Density of iron ore (hematite) in situ =5m^3/t]

Number of holes to be blasted /week = $\frac{80x3}{4.5} = 53.33 \approx 54$ numbers

Total meterage of holes =10x54 + (sub grade drilling=1x54) =594m

As per table 3.15 150mm dia. Drill machine can drill 40m/shift . Hence in a week a drill can drill 2x6x40 = 480m

Therefore two drill machines in one production bench is more than sufficient.

Overburden Bench

Height of the overburden bench = 8m

Dia. of the blast hole = 150 mm

Inclination of hole = vertical

Burden = 4.5m, Spacing = 5m.

Since a 4.6m^3 electric rope shovel can produce 2000m^3 of material and frequency of blast per week is 1 and number of rows of blast hole=3.

Length of blast per bench = $\frac{2000x6}{8x13.5} = 111.11m \approx 111m$

(Width of the blasting zone in the bench =4.5x3=13.5m)

Hence number of holes to be blasted /week= $\frac{111x3}{5} = 66.6 \approx 67$

Therefore ,total meterage of holes=8x67+ (sub grade drilling=0.5x67) = 569.5m

Hence two drill machine will be required in the overburden bench. If any drill machine goes under breakdown, the remaining 3 drill machine will cover up the minimum requirement of production of a week. Also when one drill machine will remain idle, the standby drill machine must be shifted to the second mineral bench for drilling holes.

Drilling and blasting operation is to be done in the second mineral bench for one week capacity for the sake of quality control and also keeping it as a stand-by.

Other machinery for drilling and blasting operation :

Slurry pump truck = 1

Explosive van = 1

Engineer's jeep = 1

Manpower for Drilling and Blasting Job

Drilling Manpower (per day)	Blasting Manpower
Driller = 2x4 = 8	Supervisor =1
Driller helper = 2x4 = 8	Blaster =1
Supervisor = 1x3 = 3	Blasting helper =2
Maintenance crew = 4	Explosive van driver =1
	Pump truck driver =1

The drilling and blasting operation will also be partially backed by the site engineers and managers, staffs of the site office, workshop, main office, etc.

For Production Bench

Approximate charged column = 8m
Approximate stemming height = 3m

Primer - Toe Blast (1.5 m column height, approx.)	Charge per hole =	18.75 kg
Main charge - SMS (6.5m column height, approx.)	Charge per hole (approx.) =	133.25 kg
	Total charge (approx.) =	152.00 kg

Tonnage of production /kg of charge = $\frac{8000x6}{152x2x27} = 5.847t$

For Overburden Bench

Approximate charged column = 6m
Approximate stemming column = 2.5m

Primer = Toe Blast (1m column height, approx.)	Charge per hole (approx.)	= 12.50 kg
Main charge - SMS (5m columns height, approx.)	Charge per hole (approx.)	= 102.50 kg
	Total charge (approx.)	= 115.00 kg

Cubic meter of overburden / kg of charge = $\frac{2000x6}{115x67} = 1.557m^3$

During calculating cost of drilling and blasting in the surface mining system, drilling cost accounts around 46% and the blasting cost accounts around 54% of the total cost. Explosive accounts around 3% to 8% of the total production cost per Tonne of ore.

Mine Drainage and Pumping System

Since mine is situated over the hillock ,no pump is necessary in the mine except a small pump is essential in the loading point and railway siding for proper drainage of water during monsoon. However to protect the bench crests from sliding down during heavy rain, lateral as well as inclined drains/ditches are excavated for free and controlled flowing of water through them. In the figure 4.7 (A) it has been shown that at the bottom most bench and near to the side of edges, lateral drain are located whereas on the slope of the hillock inclined flume is excavated for the said purpose. It will not only guard against the land sliding but also it will prevent ground water pollution problem.

Land Reclamation and Revegetation

It is already mentioned that the overburden will be dumped over the valley to fill it up. If the valley is filled up totally , it will be graded by the bulldozer. After filling the valley the overburden will be dumped around the outside of the periphery of the mine. When extraction of ore will be completed , the dumped overburden will be dozed and to be leveled up properly. If the valley is not filled up , first of all level up the dumped overburden over the valley and afterward spread top soil over the graded overburden as well as over the flat hillock demineralized area.

The thickness of soil depends upon the extracted area. The filled up area shall be graded in 1 in 20 with the help of bulldozer. The top soil thereafter be restored by amelioration and bio-fertilization by the agents for generation of micro-biota. After bio-reclamation the reclaimed land shall be developed into a pasture land and afforestation is to be carried out by the plantation of fast growing trees like cassia auriculata, bambusa nana, acacia nilotica, dalbergia sisoo, prosopis juliflora, etc.

For Lighting Fencing Supply of Electricity, etc.: Consult previous layout problems.

Production and Productivity

Production (given) = 8000 Tonne/day

Expected overall productivity (OMS) = 20 Tonne/man/shift

Estimated requirement of total manpower in the mine = 400 persons

NOTE: The layout which is shown with this layout problem may be applicable even for production of 5 million Tonne of ore per annum. Since the density of iron are is $5t/m^3$ (in-situ density), one bench can produce 2 to 2.5 million Tonne of iron are per annum.

LAYOUT PROBLEM – 8

An Iron ore deposit of 10m thick dipping at an angle of 45° and outcrops on the top of the hill. Overlain soil cover is 5m thick. Give layout of a quarry and requirement of machinery to produce 1000 Tonne per day of saleable iron ore. Assume any condition if necessary.

SOLUTION

Introduction

- The thickness of iron ore deposit = 10m
- The dip of the deposit =45°
- Overburden (soil) thickness =5m
- Deposit is outcropped on the top of the hill but the deposit ends at a point 10m above from the foot of the hillock.
- Deposit is quite sufficient and the life of the mine is expected around 10 years.
- Railway link is available 2 km away from the hillock.

Description of the Mining Layout

Let the selling price of the iron ore is Rs 350.00 per Tonne. So for producing 1000 Tonne of iron ore the mine management can spend around Rs 8.4 Crore for purchasing plant and machinery (from Table No. 3.28). Density of Iron ore is 5 Tonne/m^3. Hence for 1000 Tonne production per day amount of mineral excavation necessary is 200 m^3 of blasted ore (in situ) and if bulk density of ore is 4 Tonne/m^3 , 250 m^3 of blasted ore is necessary. Overburden is soil and its thickness is only 5m. Hence in the plan maximum available floor width for overburden soil is 7m and total width of the mineral bench is 21m including the width of the overburden. Since overburden is soft soil, along the slope and mechanization is not possible, the mine management decided to extract total overburden exclusively manually and the mineral bench by drilling, blasting and manual method of extraction.

Height of the overburden face = 3m.

Slope of both the overburden and mineral bench =45°

One or two teams of miners consisting of 2 to 3 members in each team will excavate the top soil with the help of hand picks and throw the spoil along slope of the hillock. The mineral bench will be drilled and blasted and several teams of miners consisting of 2 to 3 members in each team will load ore over the $1m^3$ capacity side tipping mine tubs. 14 kg/ m length rail tracks are laid over the bench, through which miners manually hauls the mine tub and unload ore over a flume. The flume is excavated from the foot of the hill to the top of the hillock by excavating soil at an inclination of 45°. At the bottom of the flume bunker is constructed (Fig 4.9). In the bunker there is the arrangement of hopper and apron conveyor for loading ore smoothly over the railway wagon.

The rail network is extended near to the loading point in which there is also the provision of shunting arrangement. Ore is directly dumped over the flume which fall under gravity and fills up the loading bunker, since the angle of inclination of the flume is more than the angle of repose of the material (37°). Special barricade is erected crosswise and in sufficient height so that the dumped soil can not jam the railway line. Besides above mine management has to construct bunker and loading arrangement at the bottom of the flume, lay 14 kg/m tracks for the mine tubs over the top of the bench, arrange electric supply line connection in the mine including procurement of necessary electric items, etc. Mine management has also to arrange a portable workshop containing all the equipment for repairing and maintenance. It should also include portable welding set, gas cutting arrangement, etc. Medical appliances for first aid, office equipment like typing machine, surveyors equipment and tools, etc. are also to be procured. A small office store, small workshop, etc. are to be constructed.

Method of Working

First of all remove all the trees and bushes from the mining area as well as from the slope (mining side) of the hillock. Grub the tree stumps manually with the help of picks or by blasting and remove them. Clean the mining site. Make the box cut by excavating trench over the soil along the strike direction. Make flume bunker and loading arrangement. Barricade the railway tracks so that overburden soil can not fall over them.

After making trench, excavate the full width of the overburden soil manually by picks and arrange to throw it at the foot of the hillock manually by the hand shovel and baskets by two teams in each shift comprising of two to three miners per team. When the overburden face will move sufficiently in the strike direction, start extraction of iron ore by drilling and blasting method up to the full width of the ore body to some distance so that tracks for the mine tub can properly be laid over that void space as shown in the (Fig 4.9). With the help of jack hammer start drilling operation for

Main Equipment

The main equipment required to be procured are as following :

Main equipment	Total number
Jack Hammer Drill	8
Air Compressors ($7m^3$/min)	2
Diesel Jeep	1
Side Tipping Mine Tube (1 m^3)	40
11 Tonne Dozer 67.5 kW	1
Centrifugal pump (50 l/s capacity, 35 kW, 440 V)	1
Truck for material transport	1
Exploder, Circuit tester, etc.	

three days production. Drilling and blasting operation is to be done in 2 lifts in a 3m bench height. Length of the drill holes are 1.5m, spacing of the holes 1.5m and burden 1.2m. A strip of 2.4m width of bench is to be blasted in one time for a 3 days production. Drilling operation with the jack hammer is selected because saleable ore is to be delivered directly from the mine and mine management has no capacity to install a costly crusher plant. The boulders sizes above + 150mm should be broken by the miners manually with the help of hammer. For loading the blasted ore, number of teams is allotted space in the bench for loading blasted ore over the mine tubs. Generally, each team is given a space around 2m to 2.5m longitudinally. The loaded mine tubs are manually transported from the face and ore is unloaded over the flume. Ore falls down by gravity into the bunker. From the bunker ore is smoothly loaded over the railway wagon with the help of apron feeder. In the fig 4.9 only one track has been shown in the face since there is no space but as the bench progress towards the footwall side one more tracks is to be installed in the face for the purpose of shunting operation. After taking the 3m slice completely, next 3m slice is to be taken below this working bench and in this manner the whole iron block will be extracted.

Drilling and Blasting

Drilling work is carried out by the jack hammers.

Length of each hole = 1.5m, Burden = 1.2m, Spacing =1.5m

Number of rows of hole per blast =2

Width of the blasting strip =2 x 1.2 =2.4m

Frequency of blasting per week =2

Density of iron ore (in-situ) =5 tonne/m^3

Density of iron ore (bulk) =4 tonne/m^3

Number of shifts: 2 production shift and one maintenance shift.

So length of blast = $\frac{1000x3}{1.5x2.4x5} = 166.66m \approx 167m$

Number of holes to be blasted for 3 days production = $\frac{167x2}{1.5} = 222.66 \approx 223$ number

On an average one jack hammer can drill 40 to 50 holes per shift. Let in this case it is only 45 numbers.

Hence in 3 days one jack hammer drill can drill

45 x 2 x 3 =270 number of holes.

Therefore one jack hammer drill is sufficient to perform this task. But it will be better if 2 jack hammer drills are operated simultaneously. One $7m^3$/min capacity mobile air compressor is sufficient to supply the required quantity of air to two drills to operate simultaneously. One more stand-by mobile air compressor is necessary to be procured. If the first air compressor comes under breakdown, the second one shall be started.

Explosives Used

Explosives may be used 80% Special Gelatine, Superdyne, Kelvex 220, Mexgel, Detamite 80, etc. which have sensitivity to No.6 detonators.

Cartridge diameter =25mm, Each cartridge weight = 140gm

Number of cartridge used per hole =3 i.e. 3 x 140 =420 gm =0.42 kg of charge.

Drilling pattern is staggered and firing pattern is In line. Electric firing is employed here with the help of multi-shot exploder (100 or 200 shot exploder).

Powder Factor = $\frac{1000}{270x0.42} = 8.81$ Tonne/kg of explosive.

For Lighting, Fencing, etc. : Consult previous layout problems

Mine Drainage and Pumping

Since mine is situated over the hillock no pump is necessary in the working places in the mine except a small capacity pump which is essential in the loading point and railway siding for proper drainage of water during monsoon. To protect the bench crests against sliding down during the heavy rain, lateral as well as inclined drains/ditches should be excavated for free flowing of water in the controlled manner. But since overburden is the top soil, land sliding as well as damaging of the flume during the monsoon is inevitable due to heavy rain fall. Hence it will be better if the mine working is suspended during heavy rain fall. The loss of production due to the temporary closure of the mine during that period shall be compensated during the normal working days.

Land Reclamation and Revegetation

After winning process of iron ore is over, the top soil shall be dozed over the void area to fill it up totally in sufficient thickness since enough soil is available. The total demineralized area as well as spoil dump yard shall be properly graded in 1 in 20 with the help of bulldozer. The graded top soil to be restored by amelioration and bio-fertilization by the agents for generation of micro-biota. After bio-reclamation is over, the reclaimed land shall be developed into a pasture land and afforestation is to be carried out by plantation of various kinds of plants (as already mentioned in the previous layout problem) and also with the flower and fruit trees to make it a very good picnic spot. It would be better if the slope area of the demineralized hillock is also sprayed with layer of soil and cultivation of grasses of varieties like cainas caian, cymbopogen citratus, bromas inermis, chloris gayana, etc. are done over that inclined area for making a very beautiful visiting place of interest.

The rail network is extended near to the loading point in which there is also the provision of shunting arrangement. Ore is directly dumped over the flume which fall under gravity and fills up the loading bunker, since the angle of inclination of the flume is more than the angle of repose of the material (37°). Special barricade is erected crosswise and in sufficient height so that the dumped soil can not jam the railway line. Besides above mine management has to construct bunker and loading arrangement at the bottom of the flume, lay 14 kg/m tracks for the mine tubs over the top of the bench, arrange electric supply line connection in the mine including procurement of necessary electric items, etc. Mine management has also to arrange a portable workshop containing all the equipment for repairing and maintenance. It should also include portable welding set, gas cutting arrangement, etc. Medical appliances for first aid, office equipment like typing machine, surveyors equipment and tools, etc. are also to be procured. A small office store, small workshop, etc. are to be constructed.

Method of Working

First of all remove all the trees and bushes from the mining area as well as from the slope (mining side) of the hillock. Grub the tree stumps manually with the help of picks or by blasting and remove them. Clean the mining site. Make the box cut by excavating trench over the soil along the strike direction. Make flume bunker and loading arrangement. Barricade the railway tracks so that overburden soil can not fall over them.

After making trench, excavate the full width of the overburden soil manually by picks and arrange to throw it at the foot of the hillock manually by the hand shovel and baskets by two teams in each shift comprising of two to three miners per team. When the overburden face will move sufficiently in the strike direction, start extraction of iron ore by drilling and blasting method up to the full width of the ore body to some distance so that tracks for the mine tub can properly be laid over that void space as shown in the (Fig 4.9). With the help of jack hammer start drilling operation for

Main Equipment

The main equipment required to be procured are as following :

Main equipment	Total number
Jack Hammer Drill	8
Air Compressors ($7m^3$/min)	2
Diesel Jeep	1
Side Tipping Mine Tube (1 m^3)	40
11 Tonne Dozer 67.5 kW	1
Centrifugal pump (50 l/s capacity, 35 kW, 440 V)	1
Truck for material transport	1
Exploder, Circuit tester, etc.	

three days production. Drilling and blasting operation is to be done in 2 lifts in a 3m bench height. Length of the drill holes are 1.5m, spacing of the holes 1.5m and burden 1.2m. A strip of 2.4m width of bench is to be blasted in one time for a 3 days production. Drilling operation with the jack hammer is selected because saleable ore is to be delivered directly from the mine and mine management has no capacity to install a costly crusher plant. The boulders sizes above + 150mm should be broken by the miners manually with the help of hammer. For loading the blasted ore, number of teams is allotted space in the bench for loading blasted ore over the mine tubs. Generally, each team is given a space around 2m to 2.5m longitudinally. The loaded mine tubs are manually transported from the face and ore is unloaded over the flume. Ore falls down by gravity into the bunker. From the bunker ore is smoothly loaded over the railway wagon with the help of apron feeder. In the fig 4.9 only one track has been shown in the face since there is no space but as the bench progress towards the footwall side one more tracks is to be installed in the face for the purpose of shunting operation. After taking the 3m slice completely, next 3m slice is to be taken below this working bench and in this manner the whole iron block will be extracted.

Drilling and Blasting

Drilling work is carried out by the jack hammers.

Length of each hole = 1.5m, Burden = 1.2m, Spacing =1.5m

Number of rows of hole per blast =2

Width of the blasting strip =2 x 1.2 =2.4m

Frequency of blasting per week =2

Density of iron ore (in-situ) =5 tonne/m^3

Density of iron ore (bulk) =4 tonne/m^3

Number of shifts: 2 production shift and one maintenance shift.

So length of blast = $\frac{1000x3}{1.5x2.4x5} = 166.66m = 167m$

Number of holes to be blasted for 3 days production = $\frac{167x2}{1.5} = 222.66 \approx 223$ number

On an average one jack hammer can drill 40 to 50 holes per shift. Let in this case it is only 45 numbers.

Hence in 3 days one jack hammer drill can drill

45 x 2 x 3 =270 number of holes.

Therefore one jack hammer drill is sufficient to perform this task. But it will be better if 2 jack hammer drills are operated simultaneously. One $7m^3$/min capacity mobile air compressor is sufficient to supply the required quantity of air to two drills to operate simultaneously. One more stand-by mobile air compressor is necessary to be procured. If the first air compressor comes under breakdown, the second one shall be started.

Explosives Used

Explosives may be used 80% Special Gelatine, Superdyne, Kelvex 220, Mexgel, Detamite 80, etc. which have sensitivity to No.6 detonators.

Cartridge diameter =25mm, Each cartridge weight = 140gm

Number of cartridge used per hole =3 i.e. 3 x 140 =420 gm =0.42 kg of charge.

Drilling pattern is staggered and firing pattern is In line. Electric firing is employed here with the help of multi-shot exploder (100 or 200 shot exploder).

Powder Factor = $\frac{1000}{270x0.42} = 8.81$ Tonne/kg of explosive.

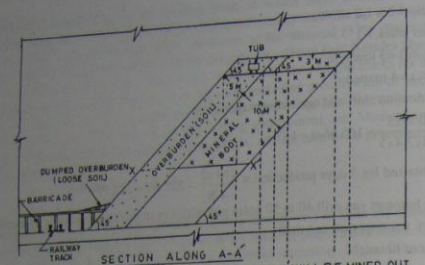
For Lighting, Fencing, etc. : Consult previous layout problems

Mine Drainage and Pumping

Since mine is situated over the hillock no pump is necessary in the working places in the mine except a small capacity pump which is essential in the loading point and railway siding for proper drainage of water during monsoon. To protect the bench crests against sliding down during the heavy rain, lateral as well as inclined drains/ditches should be excavated for free flowing of water in the controlled manner. But since overburden is the top soil, land sliding as well as damaging of the flume during the monsoon is inevitable due to heavy rain fall. Hence it will be better if the mine working is suspended during heavy rain fall. The loss of production due to the temporary closure of the mine during that period shall be compensated during the normal working days.

Land Reclamation and Revegetation

After winning process of iron ore is over, the top soil shall be dozed over the void area to fill it up totally in sufficient thickness since enough soil is available. The total demineralized area as well as spoil dump yard shall be properly graded in 1 in 20 with the help of bulldozer. The graded top soil to be restored by amelioration and bio-fertilization by the agents for generation of micro-biota. After bio-reclamation is over, the reclaimed land shall be developed into a pasture land and afforestation is to be carried out by plantation of various kinds of plants (as already mentioned in the previous layout problem) and also with the flower and fruit trees to make it a very good picnic spot. It would be better if the slope area of the demineralized hillock is also sprayed with layer of soil and cultivation of grasses of varieties like cainas cainan, cymbopogen citratus, bromas inermis, chloris gayana, etc. are done over that inclined area for making a very beautiful visiting place of interest.



(NOTE – WHEN THE WHOLE MINERAL BODY UPTO X X' WILL BE MINED OUT, THE TOTAL MINED OUT AREA WILL BE RECLAIMED WITH THE DUMPED OVERBURDEN LOOSE SOIL)

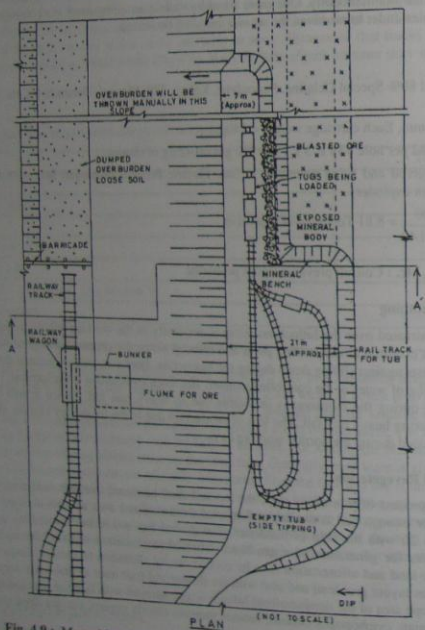


Fig. 4.9 : Manual Method of Surface Mining Operation in the Hill Top Area.

Production and Productivity

Production (given) = 1000 Tonne per day

Expected face OMS = 10 Tonne/man/shift

Expected overall OMS = 6 Tonne/man/shift

Estimated requirement of total manpower in the mine = 170 persons

LAYOUT PROBLEM 9

A very good grade patch or lenses of flat bauxite deposit of average thickness around 6m is found out on the top of the hill. Overlain overburden is 0.5 m ferruginous bauxite and over it there is a layer of 1.5m thick bauxite clay. The amount of deposit is so sufficient that the life of the quarry will be around 11 to 12 years if the production from the mine is around 1000 Tonne per day. The slope of the hill is around 45°. Give a layout of the bauxite quarry for producing 1000 Tonne per day of saleable bauxite ore. Assume any other condition if necessary.

SOLUTION

Problem 9

The layout problem of 9 (A) is more or less as similar to that of layout problem no 8 except that here the deposit is flat and situated over the hill top.

Given that the thickness of the bauxite deposit =6m

The dip of the deposit is flat

Overburden thickness is- (i) 0.5m ferruginous bauxite

(ii) 1.5m thick bauxite clay

The amount of deposit is so sufficient that the life of the quarry will be around 11 to 12 years.

Let the height of the hillock =300m

The slope of the hill is around 45°.

Let railway link is available 2 Km away from the hillock.

Around the periphery of the bauxite deposit sufficient space is available to dump overburden.

Description of the Mining Layout

Let the selling price of the bauxite ore is Rs. 183.00 per Tonne. Therefore the mine management can spend around Rs.4.39 Crore for purchasing plant and machinery (from Table no. 3.28). Density of bauxite ore is around 3 Tonne/m^3. Therefore for 1000 Tonne production per day amount of bauxite to be excavated (from in situ) is 333.33m^3 and if the bulk density of ore is around 2.2 t/m^3 then around 455 m^3 of blasted ore is to be excavated. Amount of overburden shall be excavated is (say) around 130m^3 per day.

The mine management has no capacity to install a crusher to crush the ore in saleable sizes. Since bauxite deposit is not totally bedded but vary highly both in thickness and in grade, the mine management decided to extract both the ore as well as overburden by manual method of quarrying. The overburden shall be dumped around the periphery of the mine to reuse the same in future during land reclamation. Elaborate rail track (14 kg/m length) laying arrangement is to be carried out both for the overburden (in the later stage) as well as in the mineral benches for hauling of 1m^3 capacity side tipping mine tubs. Unloading arrangement by the side tipping tubs, construction of loading point including a small bunker, railway siding, etc., shall be as the same as mentioned in the layout problem no. 8. However special barricade construction is not necessary here since overburden materials are not to be dumped over the foot of the hillock.

Main Equipment

The main equipment required to be procured for the mine are as following:

Main Equipment	Total Number
Jack hammer drill	10
Air compressors (7m^3/min capacity)	2

Table continued

Diesel jeep	1
Side tipping mine tube (1 m^3 capacity)	60
11 Tonne Dozer (67.5 kW)	1
Centrifugal pump (50 l/s capacity, 35 kW, 440 V)	1
Truck for material transport	1
For other items consult layout problem no. 8	

Method of Working

First of all remove all the trees and bushes from the mining area. Grub the tree stumps with the help of bulldozer or by blasting and remove them. Make clean the mining site. Make box cut by excavating trench over the overburden bauxite clay, ferruginous bauxite and bauxite mineral body along the strike direction manually, dump the excavated material of the surface of the trench and thereafter doze them around the periphery of the bauxite lenses by the bulldozer. Since overburden is very soft and loose, drilling blasting is not necessary. After manual method of extraction, it will be dozed and transported by the bulldozer in the first phase. First of all a 2m height overburden bench will be formed. The width of the bench will be kept around 10m.Thereafter one by one two mineral benches each having an average height of 3m and width 10m will be formed. No track laying is necessary over the overburden bench in the first phase since bulldozer will doze and transport manually excavated overburden material up to the periphery of the bauxite lenses.

But in the second phase when both the bauxite ore benches will be sufficiently progressed i.e. if at least 20m bench width is available in the bottom bench area, starting from the overburden bench , for simultaneous dumping and reclamation of the excavated area shall be done. Bulldozer will doze and grade the dumped overburden and if necessary even doze and transport back the previously dumped overburden from the periphery of the bauxite lenses. In this way simultaneous extraction of mineral as well as reclamation of void area will progress (as it has been previously seen in the layout problem no. 3). Drilling and blasting operation will be carried out in the mineral bench having a slope angle around 45°. Teams of miners consisting of 2 to 3 members in each team will load blasted ore over the 1m^3 capacity side tipping mine tubs. Generally each team is allotted with a space around 2m to 2.5m longitudinally. A strip of 2.6m width of bench is blasted in one time for a 3 days production requirement. Drilling operation with the help of jack hammers is selected since saleable ore is necessary directly from the mine. The boulders sizes above +150mm should be broken up by the miners manually with the help of hammer. The loaded mine tubs shall be manually transported from the face up to the flume where the ore will be tipped. Ore will fall down by gravity into the bunker. From the bunker ore is smoothly loaded over the railway wagon with the help of apron feeder. In every bench along with the main track there will be also provision of a shunting rail track (Fig 4.10)

Drilling and Blasting

Drilling work is carried out by the jackhammers. Length of each hole =1.5m (3m height bench shall be in 2 lifts)
Burden =1.3m. Spacing = 1.5m
Number or rows of hole per blast =2
Width of the blasting strip =2x1.3=2.6m
Number of production benches =2
Density of bauxite ore (in situ) =3 Tonne/m^3
Frequency of blasting per week = 2
Density of bauxite ore (bulk) =2.2 Tonne/m^3
Number of shifts = 2 production shift and one maintenance shift

So length of blast = $\frac{500x3}{1.5x2.6x3} = 128.2 \approx 129m$

(Since each bench has to produce 1000/2=500 Tonne of ore)

Number of holes to be blasted for 3 days production = $\frac{129x2}{1.5} = 172$ numbers

On an average one jack hammer can drill 50 holes per shift

Hence in 3 days one jack hammer drill can drill 50x2x3 =300 number of holes.

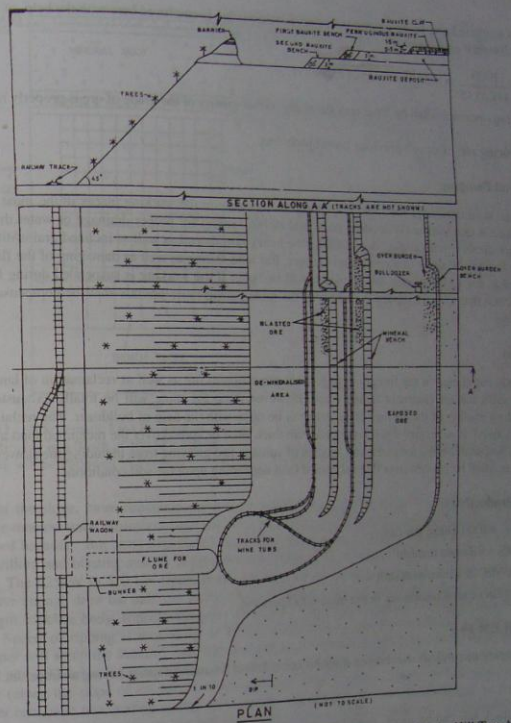


Fig. 4.10 : Manual Method of Bauxite Mining by Surface Mining Method in the Hill Top Area.

Therefore one jack hammer drill is sufficient to perform this task. For two production benches 2 jack hammer drills will be necessary. One 7m^3/min capacity mobile air compressor is sufficient to supply the required quantity of air to two drills operating simultaneously. One more stand by mobile air compressor is necessary to be procured. If the first air compressor comes under breakdown, the 2nd one shall be started.

Explosive Used

Explosive may be used 80% Special Gelatine, Superdyne,
Kelvex 220, Mexgel, Detamite 80, etc. which have sensitivity to No.6 detonators
Cartridge diameter =25mm
Amount of explosive used per hole =350 gm = 0.35 kg

Drilling pattern is staggered and firing pattern is In line. Electric firing is employed here with the help of multi-shot exploder (100 or 200 shot exploder)

Powder factor = $\frac{1000}{344 \times 0.35}$ = 8.30 Tonne/kg of explosive.

Drilling and blasting operation shall be done very carefully so that quality of the grade of ore is properly maintained.

For Lighting, Fencing etc. : Consult previous layout problems

Mine Drainage and Pumping

Since mine is situated over the hillock no pump is necessary in the working places in the mine except a small capacity pump which is essential in the loading point and railway siding for proper drainage of water during monsoon. To protect the bench crests against sliding down during the heavy rain, lateral as well as inclined drains/ditches should be excavated for free flowing of water in a controlled manner. But land sliding as well as damaging of the flume during the monsoon is inevitable due to heavy rain fall. Hence it will be better if the working is suspended during heavy rain fall. The loss of production due to the temporary closure of the mine during those periods shall be compensated during the normal working days.

Land Reclamation

It is already mentioned in the foregoing pages that extraction of ore as well as reclamation of land will go hand in hand after the sufficient development of the mine. However when the mine will be totally exhausted , final land reclamation including grading of the surface (1 in 20) shall be done with the help of bulldozer. The reclaimed land shall afterward be ameliorated. If necessary a soil layer of 10 cm thick shall be spread over the reclaimed land after bringing it from other places. Soil shall be bio-fertilized and species of suitable fast growing trees including the flower trees, various kinds of grasses, etc. shall be planted over that reclaimed land depending upon the soil condition.

Production and Productivity

Production (given) =1000 Tonne per day

Expected face OMS = 8 Tonne/man/shift

Expected overall OMS =5 Tonne/man/shift

Estimated requirement of total manpower in the mine =200 persons.

LAYOUT PROBLEM 10

Give layouts of ripping methods of overburden soil (thickness 4m) of a bauxite deposit having average thickness of 21m.

SOLUTION

The 4m thick soil can be nicely ripped by the ripper dozer combination. The schematic line diagram of cross ripping as well as ripping in one direction has been shown in the Fig 4.11 and Fig 4.12 Ripping in one direction is generally done if the soil is soft and loose but if the soil is hard then cross ripping i.e. the ripping in the same block by a number of longitudinal as well as transverse cut is practiced. In cross ripping firstly longitudinal cuts (may be transverse cut also) will be given to produce deep channels at the first instance and there after number of transverse cuts (may be longitudinal cut) will be given to produce well fragmented bulk materials. But in extremely hard rock condition pre-blasting of the rock with light charges are necessary. In this problem let consider that patches of both the hard as well as soft soils of 4 m thick are present in the mining area. Hence both the types of ripping system has to be carried out by 525 kW to 550 kW ripper dozers. The seismic wave velocity in the overburden top soil is around 1200 m/s to 1500 m/s whereas seismic wave in the bauxite is around 1400m/s to 1500 m/s. Hence both the overburden as well as bauxite mineral body is well rippable. Compressive strength of this bauxite deposit is around 100 kg/cm^2. Presence of faults, folds, dykes, etc. are absent in the overburden rock as well as in the mineral body. Hence mine management can use the ripping system for winning of both the overburden as well as the mineral body. It will yield higher productivity (since ripping and dozing operations are continuous), efficient control and maintenance both the quality and sizes of ore.

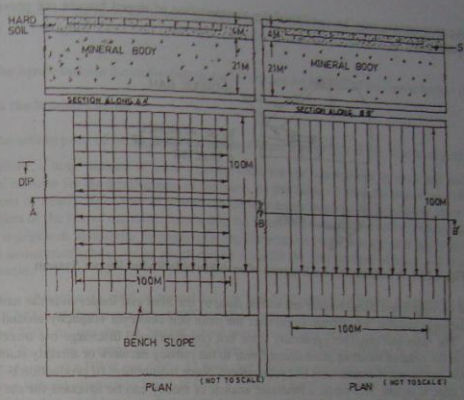


Fig. 4.11 : A Schematic Line Diagram of Cross Ripping in a Block of 100 M x 100 M (Ripping will be Followed by Dozing)

Fig. 4.12 : A Schematic Line Diagram of Cross Ripping in a Block of 100 M x 100 M (Ripping will be Followed by Dozing)

increase stability of the slope, more economic and less manpower is needed as compared to the drilling and blasting operation, very safe method since explosives need not to be handled or used and also man and machines are not exposed to any blast oriented hazards . But most important is that the system protects mine environment in a better manner as compared to the drilling and blasting system. It produces less noise, dust, vibration, etc. and can be worked out in any weather condition. The 525kW to 550 kW ripper dozer can yield on an average an output of 450 t/h. This high power rated ripper will have enough draw bar pull, can fracture large boulders, negotiate pre fractured soil and patches of higher compressive strength rock/ore body. Rippers are generally spaced in between 0.7m to 1.2 m apart depending upon the hardness of rock. Speed of ripping is around 40 to 50 m/minute. Scraper can be used along with the ripper dozer combination. Scraper will scrap the material, load into it and dump it outside the periphery of the mine boundary or in any other required place. Overburden material can be handled very efficiently with this system. Loader can be used to load, rip and doze (along the slope of the bench) overburden soil either over the dumper or belt conveyor or railway wagon or any other economic transporting system. In the same way bauxite ore can also be loaded by a tyre mounted front end loader over the transporting system like belt conveyor/railway wagons/dumpers, etc., to transport ore up to the aluminum industry/ seaport for mineral export.

The ripping block for each ripper dozer is divided into around 100m x 100 m to 200m x 200m square area. The overburden soil will be scraped, loaded, transported and dumped by the scraper outside the mining boundary (around a distance of 500 m from the central point of the deposit).

The total deposit is around 63 million Tonne (average thickness of the deposit =21m). If the production requirement per annum is 2 million Tonne , the life of the mine will be around 31 years and 6 months.

Since mineral production by ripper dozer combination is around 450 t/h, under average machine running time per day of 12 hours, on an average a ripper dozer can produce around 5400 t/day bauxite ore which is around 1.35 million Tonne per year. Therefore for 2 million Tonne production per annum number of ripper dozer requirement = 2. For overburden soil removal number of ripper dozer requirement = 2

One of the overburden ripper dozer will act as a stand by too for the mining of bauxite deposit.

The number of scraper requirement for the overburden removal is 3.

Fig 4.13 shows a schematic line diagram of a radial ripper system.

The following two options may be considered for transportation of ripped bauxite ore provided the deposit occurs at the flat surface terrain.

First Option

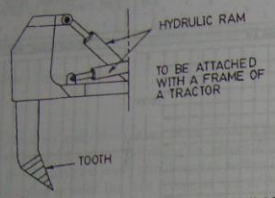


Fig. 4.13 : A Schematic Line Diagram of an Adjustable Radial Ripper System

Excavation of the ripped and dozed bauxite ore will be done by the front end loader over the mobile crusher and mobile crusher will discharge crushed ore (-60mm sizes) over the main belt conveyor (capacity around 1000 t/h) via a mobile belt conveyor. The main belt conveyor (generally cable belt conveyor) will discharge ore directly either in the aluminum industry or in the material handling plant situated near to the railway network or directly stacking yard in the sea port depending upon the site specific conditions and requirement. Since requirement of production is 2 million Tonne per annum. life of the mine is around 31.5 years, a moderate amount of money can be invested for the procurement of plant and machinery and the development of infrastructural facilities. Hence 2 numbers of front end loaders, 2 numbers of mobile crushing units (of capacity 180 to 200 m^3/h) along with 2 mobile belt conveyor each of capacity around 500 t/h and one main cable belt conveyor 1000 t/h capacity are to be procured for the transportation of bauxite ore to the required destination.

But installation of a long length cable belt conveyor of capacity 1000 t/h is very expensive equipment nowadays. Therefore, if condition permits the 2nd option is the best and cheapest mode of transportation.

Second Option

Excavation of dozed bauxite ore will be carried out by the 6.84 m^3 bucket capacity front end loader over the 180 to 200 m^3/h capacity mobile crusher and the mobile crusher will discharge crushed ore (-60 mm sizes) over the mobile belt conveyor of capacity around 500 t/h. The mobile belt conveyor will discharge ore into the railway wagon. Here railway network facilities are extended up to the mine. The rail transport system is not only the cheap, flexible, highly productive but also it will assist the development of the surrounding places because of the growing of network of cheap railway communication facilities for the local people.

LAYOUT PROBLEM-11

A lignite deposit of average thickness 20 m occurred on a flat surface terrain. The deposit is flat and the overlain overburden (soil and loose sand) is also 20 m thick. The total reserve of the lignite is around 100 million Tonne. Give a suitable surface mining layout to extract around 3 million Tonne of lignite per annum. Assume any other conditions if necessary.

SOLUTION

Introduction

Given that the thickness of the lignite deposit = 20m
Dip of the deposit =Flat
Thickness of overburden (soil and loose sand) =20 m
Production requirement=3 million Tonne/ annum i.e. 12000t/day (approx)
There is no presence of any geological disturbances like fold, faults, etc.
Let the railway link is available at a distance of 4 km away form the mine.

In the outside of the mine sufficient space is available to dump the overburden till enough space is obtained for reclamation of the de-lignite area. The spoil bank capacity is around 10 million m^3 . Dumping height of the spoil is around 25 m.

Let length of the lignite body in the strike direction =2778m and the breadth of the lignite body =1500 m

Description of the Mining Layout

Let the selling price of the lignite is around Rs.474 per Tonne. Life of the mine is around 33.33 years.

Therefore, a huge amount of money can be spent for the procurement of plant and machinery and also heavy investment can be done for the development of the mine. As per Table - 3.28 in the first phase (10 years) of the life of the mine ,an amount of nearly 205 Crore of rupees can be very easily invested for purchasing plant and machinery. Taking due consideration of the life of the mine, forecasting the future price rise of ,the heavy mining equipment and lignite, life of the mining equipment, flexibility of production requirement, etc. mine management has to come under a decision for the maximum investment (phase wise) which can be done for the procurement of plant and machinery and also for the mine development.

Density of lignite (in situ) is around 1.2 t/m^3 and bulk density of lignite is around 0.8 t/m^3. Therefore for the production of 12000 t of lignite per day, 15000 m^3 of excavated lignite has to be handled. Amount of overburden is to be handled around 8500 m^3 per day. A bucket wheel excavator having bucket capacity 700 liters has theoretical capacity around 2730 m^3/h. On an average this type 700 liters BWE can produce 2000 m^3/h of bulk material. Therefore, if the average working time of a BWE per day is 12 hour, one BWE can excavate around 24000 m^3 of material. The height of the each overburden and production bench is 24m. Keeping of stand by machine is not possible because of the expensive equipment. Taking due consideration of standardization for overburden bench one 700 liter BWE with deep cut facilities and for lignite bench also one 700 liter BWE with deep cut facilities has been selected. The overburden BWE is combined with a spreader having capacity 1000 m^3/h with a discharge boom length around 100m to 125m. The lignite will be excavated by the BWE and directly discharge over the railway wagon as shown in the Fig 4.14. Since ground is flat and rail network is available within 4 km from the mining site, the railway facility is directly linked inside the mine for easy and cheap transportation of lignite to the thermal power station. Railway line is laid along the strike direction of the lignite body.

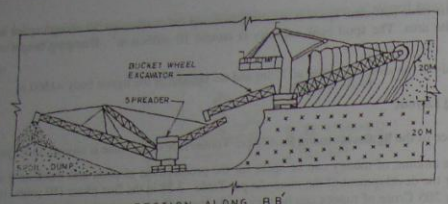
The height of both the lignite and overburden bench is 20 m each.

The breadth of the overburden bench =40 m

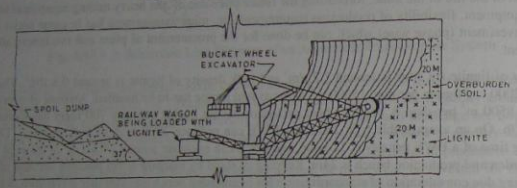
Angle of slopes of both the benches are around 45° in hemispherical shape as shown in the (Fig. 4.14). Edges of any bench will also not be very smooth and they will be looked corrugated since excavation of overburden soil/lignite is done by the bucket of the BWE (terrace cut). As the winning of lignite is in progress , the railway line will be also extended up to the extreme end of the boundary in the strike direction i.e. up to 2778 m at the time of taking first slice of the lignite of width around 35 m. Thereafter the railway line will be shifted along the direction of the breadth of the lignite body every after taking one slice. The frequency of railway line shifting will take nearly more than 9 months per shifting since the yearly production of lignite is only 3 million Tonne.

Method of Working

First of all remove phase wise all the trees and bushes from the mining area. Grub the tree stumps with the help of bulldozer or by drilling and blasting operation and thereafter remove them to clean the mining site. Make box cut by a hydraulic shovel and rear discharge dumpers combination. First make sufficient room over the exposed lignite to install BWE for overburden and spreader. Start advancing the overburden face in the strike direction and at the same time give box cutting to the lignite. Excavate and transport lignite by the same hydraulic shovel and dumper combination. When sufficient room will be available , lay railway track and install BWE meant for lignite. The lignite BWE will excavate around 35 m width slice and will advance along the strike direction up to the end of the boundary. Since overburden removal by overburden BWE will be faster, the overburden BWE after finishing excavation of loose soil and sand will start retreating by excavating another slice of around 35m from the boundary end towards the initial starting point. As soon as extraction of the first slice by the lignite BWE will be completed, shift the railway line around 35 m towards the bench. In this manner complete taking of slice after slice of overburden as well as lignite. The overburden which will be dumped in the dumping place outside the excavating area during making box cut, making room for BWE and spreader



SECTION ALONG BB'



SECTION ALONG A A' (OVERBURDEN REMOVAL NOT SHOWN)

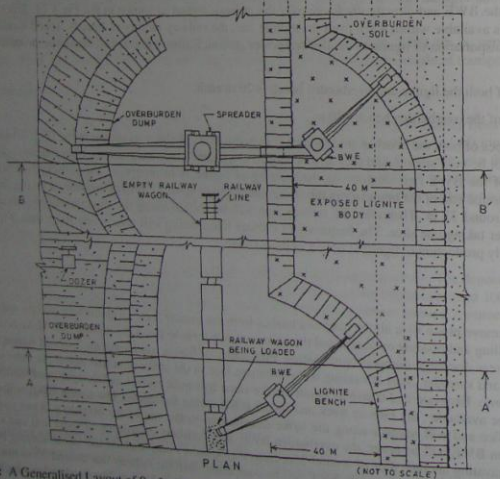


Fig. 4.14 : A Generalised Layout of Surface Mining Operation by Bucket Wheel Excavaour and Spreader Combination (in a Lignite Deposit)

and also during extraction of first and second slice will be dozed back inside the excavated de-lignite area and shall be properly reclaimed. Overburden material of subsequent slices will be directly dumped over the de-lignite area with the help of spreader. However during excavating the overburden at the extreme end of the boundary, some amount of overburden will be dumped again over the surface which will also require to be dozed into the de- lignite area later on. Since the shovels and dumpers will be used only for making box cut, it will be better if they are hired form the mining company for a short period. In both the overburden and lignite benches terrace cut will be practiced along with the full block method of working. The BWE meant for overburden excavates and discharges overburden material over the spreader which in turn dumps the overburden by spreading and slewing (Fig 4.14). Later on dozer will doze and grade the dumped material at an gradient of 1 in 20. The angle of inclination of spoil dump slope is around 30°. If patches of hard rock is present in the soil , it will be blasted either by Toe blast, O.C.G. Bharat Toe, Mex-Blast, etc.

Main Equipment

The main equipment which are to be procured are as following-

Equipment	Number
700 liters Bucket Wheel Excavator	2
Spreader (1000 m^3/h capacity)	1
21 Tonne 135 kW Dozer	2
Diesel Jeep	3
Trucks for material transport	2
Pumps (300l/s capacity)	2
Maintenance van	1
5 MVA Transformer	3

Other Equipment to be Procured

- Mobile Crane.
- Welding truck.
- Giraffe truck.
- Cable reel truck.
- Steam cleaning machine.
- Ambulance van, equipment and medical appliances for a standard hospital.
- Computer centre along with various kinds of software.
- 100 mm dia. electric blast hole drill -1 no.
- Portable workshop containing all the equipment for repairing and maintenance including portable electric welding set, gas cutting arrangement, etc.
- Main Workshop-It includes component shop, auxiliary equipment shop, machine shop, structural and welding shop, electrical repairing shop, engine shop, light vehicle shop, carpentry and paint shop. In the workshop besides various common equipment, E.O.T. crane, fork lifter, floor mounted hydraulic crane, etc. shall be provided.
- Exploder, circuit testers, etc.
- Safety appliances and equipment
- Electrical equipment in the sub-station like insulators, conductors, transformer, oil circuit breaker, carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, etc.

For Fencing and Lighting -Consult layout problem - 4.

For Supply of Electricity- Consult annexure - 3.1t should be noted that the minimum voltage required for running the motors of the BWE and spreader is 3.3 KV.

4. Water Jet Cutting

In this method a continuous high pressure water jet of a diameter less than 1 mm impulse at very high speed by means of a pipe specially made up of synthetic sapphire with an orifice of 0.2mm to 1 mm is injected into the granite block. This system is capable of cutting a slot of 50 mm up to a depth of 4.5 m and 5.5 m in length in a cycle of operation. However in this method the life of the pipe line is decreased and the depth of cut will be reduced.

Layout Problem-13

A flat high grade coking coal seam of 10m thick occurs at a depth of 25m from the surface. Draw a layout of the surface mine to produce 1.25 Million Tonne of coal per annum. The layout shall also show the land reclamation and revegetation program.

Solution

Thickness of coal seam : 10m

Thickness of overburden: 25m

Let the following points are, considered.

Deposit is quite sufficient and the life of the mine shall last around 11 - 12 years

Railway link is 1.5 Km away

Description of Mining Layout

If number of working days are considered 250 days, production per day = 1250000/250 = 5000 Tonne of coal.

For such an amount of production one coal bench is sufficient.

Height of the coal bench : 10m

Slope of the coal bench :65^0

No. of overburden bench is also one.

Height of the overburden bench = 25 - 2 = 23m

(Note: 2m is the thickness of the top soil. Top soil is excavated, dozed and stacked into the nearby stacking place for future use during the land reclamation).

Coal transported from the mine shall be dumped over the bunker of the coal handling plant (and also over the dump yard if necessary). From the coal handling plant, coal shall be loaded directly over-the railway wagon. The stacked coal shall be loaded over the railway wagon by a Front End Loader. The coal bench shall be winned by electric shovel - rear discharge dumper combination whereas overburden bench shall be excavated and side-casted by a dragline. From the above mentioned problem it is very clear that the stripping ratio is 1.85:1 m^3/t. So amount of overburden is to be excavated per year is 1.25x1.85 = 2.31 million cubic meter. Since the coal deposit is flat and overburden thickness is 25m, a Dragline of 16m^3 bucket capacity and 96m boom length is more than enough for excavating the overburden rock.

The Fig.4.16 shows the simple side casting system by a Dragline. In this figure the dragline is installed over the top soil free overburden rock. For exposing coal, Dragline excavates overburden rock-over the coal seam and dump it over the spoil bench. Considering the selling price of the high grade coking coal is approximately Rs 2000/- per Tonne a maximum Rs 360.00 Crore can be spent for the production of 5000 Tonne of coal (From Table No.3.28) for the procurement of plant and machinery. Hence there will not be much of difficulties to procure a 15m^3 bucket capacity Dragline, Shovel, Rear Discharge Dumpers, Dozers, Motor graders, Scrapers, and other required mining equipment.

Main Equipment

From the Table No.3.17 one 10m^3 electric Rope Shovel with five 85 Tonne Rear Discharge Dumper combination can produce very easily 5000 Tonne of coal in a day. During the surface mining operation, if the layout-is well planned and maintained, the same dragline may be used for the production of coal in case the shovel remains under breakdown.

So the following main equipment required to be procured:

Main equipment	Total number
15m^3 bucket capacity Dragline	1
10m^3 bucket capacity electric Rope Shovel	1
Drill machine (DTH for blast hole dia. 311 mm and 250mm)	1 each
85 Tonne Rear Discharge Dumper	6
Bulldozer 340 kW	2
Bulldozer 140 kW	2
Scraper	1
Motor grader	1
Compactor	1
Maintenance van	1
Diesel jeep	2
Fuel truck	1
Explosive van	1
Water sprinkler (28,000 liter capacity)	1
Pump 80l/s , 112kW, 550V	2
Transformer	As per the requirement

Other Equipment to be Procured

- Coal handling plant
- Air compressors (7 - 9m^3/minute capacity) .Number depends upon the number of jack hammer drill machine and other compressed air operated drill machine and equipment used.
- Jack hammer drills - 3 Nos.
- Portable workshop containing all the equipment for repairing including portable electric welding set, gas cutting.
- Electrical equipment in the sub-station like insulator, conductors, transformers, oil circuit breaker, carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays, control equipment, etc. should be provided. Flexible cables, for various equipment, exploders, circuit testers, etc. shall also be provided.
- Safety appliances and equipment.
- Equipment and medical appliances for first aid
- Office equipment e.g. computers along with its printers , various software and other accessories Xerox machine, Cyclostyle machine, blue printing machine, surveyor' s equipment and tools, various monitoring equipment ,etc.
- Mobile crane
- Welding truck
- Giraffe truck
- Cable reel truck
- Tyre handler
- Power sweeper
- Steam cleaning machine
- Wheel motor dolly
- Computer along with various accessories-and software for computer centre.
- The main workshop. It includes truck shop, tyre shop, component shop, auxiliary equipment shop, machine shop, structural and welding shop, electrical repairing shop, engine shop, carpentry and paint shop. In the workshop besides various equipment, EOT crane, fork lifter, floor mounted hydraulic crane, etc. shall also be provided.

Method of Working

First of all remove any trees or bushes present over the site of the quarry, grub the tree stumps either with the help of a bulldozer or simply by drilling and blasting operation. Then top soil shall be removed and stacked by the scraper to a specified area and stacking of top soil shall be done in the chronological sequence. Mining site shall be made clean and a box cut shall be made over the overburden rock along the longitudinal direction near the boundary of the mine by drilling and blasting method. Mucking and side casting of the rock shall be done by the dragline whereas mucking and transporting of coal shall be done by the shovel and dumper combination (Fig. 4.16).

When sufficient bench width shall be available the winning of overburden shall be carried out in full swing by the dragline. But until and unless sufficient amount of de-coaled area is available the overburden material shall be stacked over the surface. When sufficient amount of de-coaled area is available, the overburden material shall be excavated and side casted directly over the said de-coaled area. At the same time previously stacked overburden material shall also be rehandled and dumped judiciously over the said de-coaled area. The minimum coal bench width and length shall be 60m and 100m respectively. For proper utilization of the 10m^3 bucket capacity electric Rope Shovel, drilling and blasting operation shall be carried out in such a manner and in such away that a width of 24m face is always available. A ramp with a gradient of 1 in 10 and around 6 to 7 times width of the maximum width of the dumper ,shall be excavated to reach the coal bench.

After finishing of the 100m slice all along the strike length (longitudinal length of the mining property) the next 100m slice (along the transverse length of the mining property) shall be excavated in the same manner. By taking successive slices in the said manner, the whole mining property shall be excavated out within 12 years. Floor of both the overburden and coal benches are inclined at an inclination of 1 in 20 towards the extraction side and with the help of very good drainage system water straightway directed towards the sump during monsoon. If necessary right from the surface up to the sump drains and ditches shall be formed for the free flowing of water.

Drilling and Blasting

(a) Coal bench

Drill machine selected here is the rotary percussive drill machine to drill holes of diameter 250mm (from Table 3.13)

Height of the coal bench	: 10m
Spacing of blast-holes	: 7m
Burden	: 6m
Number of rows per blast	: 4
Width of the blast	: 4 x 6 = 24m
Length of blast	: 100m

No. of holes to be blasted/week = $\frac{100x4}{7} = 57$

So total meterage of holes to be drilled in the coal bench = 57 x 10 +.(sub-grade drilling = 0.5 x 57) = 598.5m.
This meterage of hole length can be drilled very easily by a DTH drill machine in 6 days.

b) Overburden Bench

Drill machine selected here is the rotary percussive drill machine to drill holes of diameter 311mm (from Table : 3.13)

Height of the overburden bench	: 23m
Spacing of blast holes,	: 10m
Burden	: 8m
No. of rows per blast	: 3
Width of the blast	: 3 x 8 = 24m
Length of the blast	: 100m

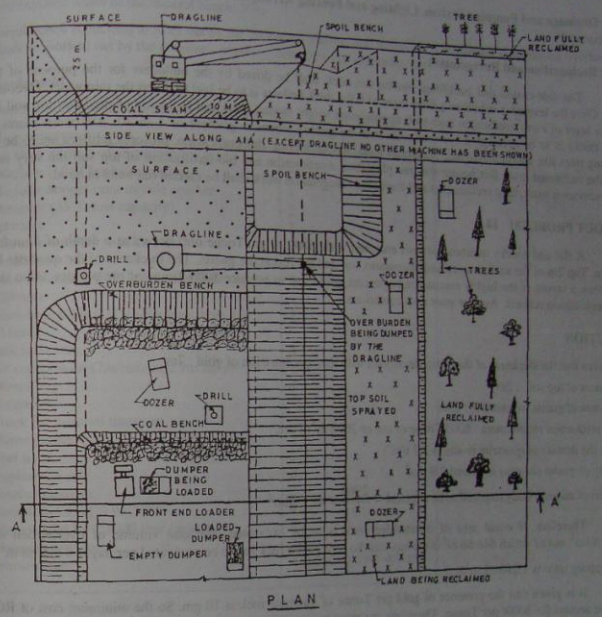


Fig. 4.16 : A generalized layout of surface mining system operated by front end loader – dumper and dragline combination

No. of holes to be blasted/week = $\frac{100x3}{10} = 30$

So total meterage of holes to be drilled in the overburden bench = 30 x 23 + (sub-grade drilling = 1x30) = 705m
This meterage of hole length can be drilled very easily by a DTH drill machine in 6 days. In both the cases

Pattern of blast-holes	: Multiple 'V' cut
Explosive used	: O.G..G. booster and aluminized slurry explosives
Initiation.	: Both down-line and trunk-lines of cordtex detonating fuse and 6 to 8 number delays are used.

Production and Productivity

Production (given) = 5000 Tonne/day
Expected overall OMS = 25 Tonne/man/shift
Estimated total manpower required = 200 persons.

Mine Drainage and Pumping System, Lighting and Fencing Arrangements, Supply of Electricity: Consult layout problem no.2

Land Reclamation and Revegetation

The side-casted and backfilled overburden rock shall be dozed by the bulldozer for the purpose of leveling work. Over the leveled overburden a filler layer of 0.8m thick ballasts is to be laid to divert the rain water. Over the filler layer a layer of earth overburden (sub-soil) approximate height around 1.5m and at the top of the latter top soil (around 0.5m thick) is to be sprayed. Layers of soils shall be graded around a gradient of 1 in 20. Afterward plants of fast growing trees like cassia auricqlata, bambousa nana, acacia nilotica, dalbergia sisoo, prosopis juliflora are to be planted over the reclaimed land. But before the tree plantation amelioration and bio-fertilization of top soil are very essential. The excavation work, land reclamation and thereafter revegetation work shall be carried out hand in hand.

LAYOUT PROBLEM 14

A flat and blocky quartzite deposit enriched with gold (10 gm/Tonne ore) occurs at a depth of 25m from the surface. Top 2m of the surface overburden contains soil and the rest is the gneiss. The thickness of the quartzite block is 1m. Draw a layout of the highly mechanized opencast gold mine to produce 8000 Tonne of ore per day. Also show the land reclamation scheme. Assume your own conditions.

SOLUTION

It is given that the thickness of the quartzite deposit with the concentration of gold : 7m

Thickness of top soil : 2m

Thickness of gneiss overburden : 23m

Daily production requirement : 8000 Tonne of ore or 2693 m^3 of ore.

(Since the density of quartzite in-situ is 2.7 t/m^3)

Density of gneiss (in-situ) is around 2.9 t/m^3

Amount of mineral body required to be exposed = $\frac{2963}{7} = 423.28m^2$

Therefore, if equal area of overburden rock is to be excavated ,the total volume of overburden shall be 10582.14m^3, out of which 846.56 m^3 is the top soil. So neat gneiss rock has to be excavated per day is 9735.58 m^3.

So stripping ratio is 1.216 : 1 m^3/t

It is given that the presence of gold per Tonne of quartzite rock is 10 gm. So the minimum cost of ROM ore shall be around Rs 3000/ per Tonne. Therefore, the mine management can spend nearly Rs 816.00 Crore for purchasing the plants and machineries for the said opencast mine. The total mine area is divided equally into 2 blocks viz. Block-I and Block-II. Both the blocks are worked out in the same manner (Fig. 4.17). The layout shows the simultaneous production and land reclamation process and the system is a semi-continuous process. If township is located within 2.Km distance, a gneiss crushing plant may be installed near the dump yard. The crushed gneiss may be used as aggregate and chips for road construction. The mine owner may earn a huge amount of revenue out of this. The well crushed ROM is directly transported to the milling plant located 2Km away from the mine for the extraction of native gold from the quartzite.

Description of Mining Layout

Each block is designed to produce minimum 4000 Tonne of ore per day. For the production of 8000 Tonne of ore, one block is sufficient but for the sake of quality control another block is constructed side by side. The each block is designed in the following way:-

Height of the production bench: 7m

Slope of the production bench: 70° (from horizontal) 2m thick top soil shall be scrapped by the scraper and it will then be stacked in a suitable stockyard for future use. So remaining 23m gneiss overburden shall constitute into only a single bench.

The slope of the overburden bench: 10° (from horizontal)

Minimum exposed width of the mineral bench: 30m

The mineral bench is decided to work out by the backhoe - crusher - belt conveyor combination Whereas the overburden bench shall be worked out by the dragline - crusher – belt conveyor - spreader combination.

Main Equipment

The main equipment to be procured are as following :

Main equipment	Total number
Backhoe (4m^3 bucket capacity)	2
Dragline (10m^3 bucket capacity)	2
Modular hopper cum mobile crusher	4
Dumpers (35 Tonne capacity)	2
Spreader	2
Belt conveyor (in smaller units)	20 Km
Bulldozer	7
Blast-hole drills (DTH)(For blast-hole dia. 250mm)	2
Blast-hole drills (DTH)(For blast-hole dia. 311mm)	2
Jack hammer	8
Air compressor (7m^3/min and 9m^3/min)	1 each
Diesel jeep	4
Scraper	2
Truck for material transport	2
Motor grader	2
Fuel truck	2
Explosive van	2
Road compactor	2
Pumps 80 l/s, 112kW ,550V	4
Water sprinkler (28000 liter capacity)	2
Maintenance van	2
Transformer	As per the power demand.

Other Plants and Equipment to be Procured

- Ore handling plant
- Gold extracting milling and processing plant
- Portable workshop containing all the equipment for repairing including portable electric welding set, gas cutting arrangement, etc.
- Electrical equipment in the sub-station like insulators, conductors, circuit breakers, carrier current equipment, protective fuses and relays, control cable and conduit system, switch boards containing meters, relays control equipment, etc. Flexible cables for various equipment , exploders, circuit testers, etc.
- Safety appliances and equipment.
- Equipment and medical appliances for first aid
- Office equipment like computers along with its printers , various software and other accessories, Xerox machine, cyclostyle machine, blue printing machine, surveyor's equipment and tools, various monitoring equipment, etc.
- Mobile crane
- Welding truck
- Giraffe truck

- Cable reel truck
- Tyre handler
- Power sweeper
- Steam cleaning machine
- Wheel motor dolly
- Computer along with the various kinds of software for computer centre.
- Mine workshop. It includes truck shop, tyre shop ,component shop, auxiliary equipment shop, machine shop, structural and welding shop, electrical repairing shop, engine shop, light vehicle shop, carpentry and paint shop. In the workshop besides various equipment, EOT crane, fork-lifter, floor mounted hydraulic crane, etc. shall also be provided.

Method of Working

First of all remove any trees and bushes present over the site of the quarry, grub the tree stumps either with the help of bulldozer or simply by drilling and blasting method. Scrap the top soil by the scraper and dozer combination and dump the top soil in a suitable stack-yard in chronological sequence. Make clean the mining site. The total mining area shall be divided into 2 blocks (Block I and Block-II) and a box cut shall be given near to the end of the boundary of the mine with the help of drilling and blasting method (Fig. 4.17). Mucking and side casting of over burden shall be done with the help of dragline whereas mucking and transportation of ore shall be done with the combination of backhoe and dumpers. When sufficient exposed width in the mineral bench (at least 35m) is available , both the winning of mineral and excavation of overburden shall be carried out in full swing and simultaneously. For proper utilization of $4m^3$ bucket capacity backhoe and $10m^3$ bucket capacity dragline, drilling and blasting operation shall be carried out in such a manner and in such a width that a 15m to 16m and 20m to 24m face widths are always available for excavation by the backhoe and dragline respectively. The excavated ore by the backhoe shall be dumped over the hopper of the mobile crusher unit where the ore shall be sized into -150mm size and the crushed ore shall be transported directly to the ore handling plant of the milling cum processing plant with the help of a number of belt conveyors. The excavated overburden rock shall be dumped over the hopper of the crusher unit where it shall be sized into -200 mm. Initially the crushed rock shall directly be transported over the dump yard and nicely be spreaded and stacked by the stacker. But afterward as the sufficient de-mineralized area is available ,the crushed overburden gneiss shall be used for backfilling of the de-mineralized area. This will minimize the hauling distance of the overburden as well as it will eliminate rehandling of the said overburden. At the finishing point of the pit the dumped overburden at the outside the mine shall be re-handled and dumped over the de-mineralized area. Floor of the overburden bench shall be inclined at an inclination of 1 in 20 towards the extraction side and during excavation ,the floor of the mineral bench shall be inclined towards the sump so that water during the monsoon season 'can directly flow into the said sump. If necessary right from the surface up to the sump drains and ditches are to be formed for free flowing of water.

Drilling and Blasting in Each Block

a) Mineral Bench

Height of the mineral bench	:	1m
Spacing	:	6m
Burden	:	5.5m
Angle of inclination	:	10^0 from vertical

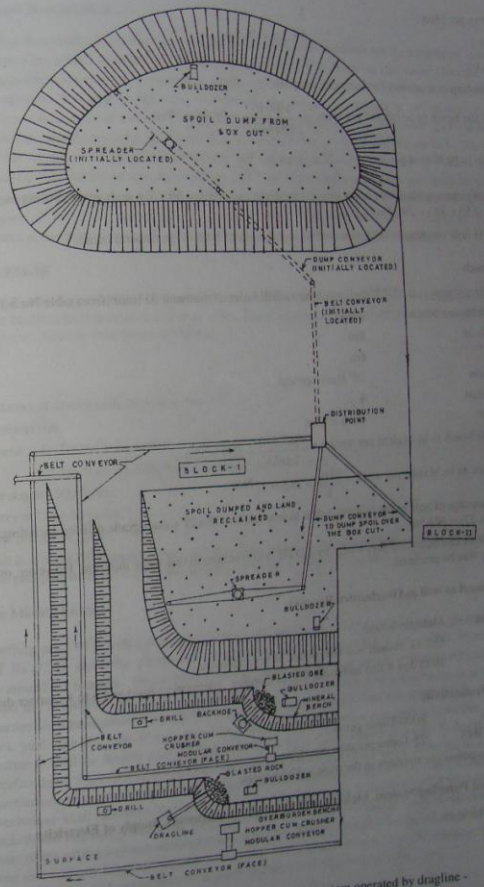


Fig. 4.17 : A generalized layout of surface mining system operated by dragline - mobile belt conveyor –spreader combination

Number of rows per blast : 3

Width of blast : $3 \times 5.5 = 16.5m$

Number of face : 1

Drilling and blasting is necessary for one week duration:

Total length of the bench to be blasted per week $= \frac{4000 \times 6}{16.5 \times 7 \times 2.7} = 76.96 \approx 77m$

Number of holes to be blasted per week $= \frac{77 \times 3}{6} = 38.5 \approx 39$

So total length of meterage blast hole in the mineral bench in one week = 39 x 7 + (sub-grade drilling including the factor of inclination = 0.50 x 39) = 292.5m

Hence, one DTH drill machine is sufficient to drill such length of hole in the mineral bench.

Overburden Bench

Drill machine selected here is the DTH drill machine to drill holes of diameter 311mm (from table No.3.13)

Height of the overburden bench	:	23m
Spacing of blast hole	:	8m
Burden	:	6m
Angle of inclination	:	10^0 from vertical
No. of rows per blast	:	4
Width of the blast	:	4 x 6 = 24m

Total length of the bench to be blasted per week : $= \frac{4867.5 \times 6}{24 \times 23} = 52.9 = 53m$

So number of holes, to be blasted per week $= \frac{53 \times 4}{8} = 26.5 \approx 27$ Nos

Therefore total meterage of hole in the mineral bench in one week = 23 x27 + (sub-grade drilling including the factor of inclination = 1.5 x 27) = 661.5m.

This meterage of hole can be drilled very easily by a DTH drill machine in one week duration. However, one standby drill machine may also be procured.

For both the Mineral as well as Overburden Benches

Pattern of blasthole	:	Multiple 'V' cut
Explosive used	:	O.C.G. booster and aluminized slurry explosive.
Initiation	:	Both down-line and trunk-line of Cordtex (detonating fuse) and 6 to 8 number delays are used.

Production and Productivity

Production (given)	:	8000 Tonne per day
Expected overall OMS	:	25 Tonne per man per shift

Estimated total manpower requirement in the mine = 320 persons

Mine Drainage and Pumping System, Lighting and Fencing Arrangements, Supply of Electricity :

Consult layout problem no.2

Land Reclamation and Revegetation

The backfilled overburden rock (here given) shall be dozed by the bulldozer for the purpose of leveling work. Over the leveled overburden a filler layer of 0.8m thick ballasts is to be laid to divert the rain water. Over the filler layer, a layer of earth overburden (sub-soil) of approximate height around 1.5m and at the top of the latter top soil (around 0.5m thick) is to be sprayed. The land reclamation and thereafter revegetation work shall be carried out simultaneously with the production process. Layers of soils shall be graded around a gradient 1 in 20. Afterward plants of fast growing trees like cassia auriculata, bambousa nana, acacia nilotica, dalbergia sisoo, prosopis juliflora are to be planted over the reclaimed land. But before the tree plantation amelioration and bio-fertilization of top soil are very essential. It shall be noted that after a sufficient amount of demineralization and land reclamation work, top soil shall be scrapped and brought straightway over the backfilled and leveled gneiss overburden rock and soil shall be sprayed thereafter in chronological sequence. It will help damaging of very less amount of surface land (due to stacking), very less soil degradation, eliminates rehandling of soil and decrease the cost of overall land reclamation process.

LAYOUT PROBLEM-15

A flat phosphorite deposit of 3m thick occurs in a patch on the hill top area. The overburden contains 2m top soil followed by 3m medium hard limestone and marl strata. Draw a layout of the surface mine showing land reclamation and revegetation system. Assume your own conditions.

SOLUTION

Given that the thickness of phosphorite deposit = 3m

The dip of the deposit is flat

Overburden thickness: i) 2m thick top soil

ii) 3m thick medium hard limestone and marl strata

The production per day = 1000 Tonne

The life of the quarry = 11 to 12 years

Let the height of the hillock = 100m ,

The slope of the hill is around 20^0 from horizontal. Let the railway link is available 1 km away from the hillock. Around the periphery of the phosphorite deposit sufficient space is available to dump the top soil.

Description of the Mining Layout

Let the selling price of the phosphorite ore is Rs 1572 per Tonne. Therefore, mine management can spend a maximum around Rs 37.73 crore for purchasing plant and machinery (from Table No. 3.28). Density of friable phosphorite ore is around 2.5 tonne/m^3. So for 1000 Tonne of production per day amount of phosphorite to be excavated (from in-situ) is 400 m^3. If the bulk density of ore is 2 tonne/m^3 , around 500 m^3(swell volume) ore is to be excavated. Amount of overburden shall be excavated around 833 m^3 per day (swell volume) .Top soil shall be scraped and stacked in the nearby stack yard with the help of a scraper. The medium hard limestone and marl overburden rock shall be blasted and thereafter it shall be loaded onto 25 Tonne dumpers by a 2.5m^3 bucket capacity hydraulic shovel. The dumper shall thereafter dump the overburden spoil over the valley of the hillock (Fig. 4.18). Since the transport distance is very less the combination of excavation and transporting mode shall be one hydraulic shovel and 2 dumpers. This type of valley filling technique shall minimize the cost of transport and also cause beautification of the hillock after the process of land reclamation and revegetation. For the purpose of surface run-off, drainage channels shall be constructed around the perimeter of the valley and also artificial drainage system shall be provided for the drainage of groundwater. These shall be done to avoid any landslide, surface erosion and detrimental effects on the fill stability. The valley fill method increases the safety standard and reduces the dumper operation and maintenance cost.

Main Equipment

The main equipment required to be procured are as following :

Main equipment	Total number
Scraper	1
Wagon drill	2
Jack hammer	2
Air compressor ($9m^3$/min capacity)	2
Diesel jeep	2
Fuel truck	1
Tractor dozer	2
$2.5m^3$ bucket capacity hydraulic shovel	2
Rear Discharge dumper (25 Tonne capacity)	8
Motor grader	1
Compactor	1
Centrifugal pump (50 l/s capacity, 35kW ,440V)	1
Exploder, circuit tester ,etc.	3 each
Explosive van	1
Water sprinkler	1
Maintenance van	1
Transformers ,other electrical items and other eqdipment to be procured - Consult previous layout problems.	

Method of Working

First of all remove all the trees and bushes from the mining area. Grub the tree stumps with the help of bulldozer or by blasting and remove them. Make clean the mining site. Make box cut by excavating trench over the overburden top soil, limestone and marl strata and phospjhorite mineral body along the strike direction with the help of hydraulic shovel and rear discharge dumper combination. The soil later on shall be scrapped by the scraper and shall be dumped over the stack yard in chronological sequence outside the phosphorite lenses (Fig.4.18).

Drilling and blasting shall be done over the limestone and marl overburden bench with the help of wagon drill. The detail regarding the drilling and blasting is as follows:

Length of the hole : 3.2m
Burden : 4m
Spacing : 4m
Diameter of blasthole : 100mm
No. of rows of holes per blast : 2
Width of the blasting strip : 8m
No. of shifts : 2 production and 1 maintenance shift
So length of blast per week $\frac{400x6}{3x8} = 100m$

Number of holes to be blasted per week : $\frac{100 \times 2}{4} = 50$

The length of blast hole = 50 x 3.2 = 160m

A wagon drill can drill very easily this 160m length of hole in a week. However, one more wagon drill machine shall be kept ready as a standby.

Pattern of blast hole = Multiple 'V' cut

Explosive used : 40% OCG and 60% site mixed slurry explosive.

Charge per hole : 7 kg (approx)

Initiation: Both down-line and trunk-line of Cordtex detonating fuse and 6 to 10 number delays shall be used. Loose phosphorite ore is excavated directly by the hydraulic shovel and loaded over the 25 Tonne rear discharge dumper. The dumpers transport the ore and dump over the stack yard located 1 Km away from the hillock from where the ore is transported by rail to the required destination. During excavation of phosphorite ore by the hydraulic shovel, dozer shall be utilized for its assistance if necessary. In case of presence of hard patches in the phosphorite ore block, drilling and blasting may be practiced (if it is necessary). Jack hammer shall be used for the secondary blasting.

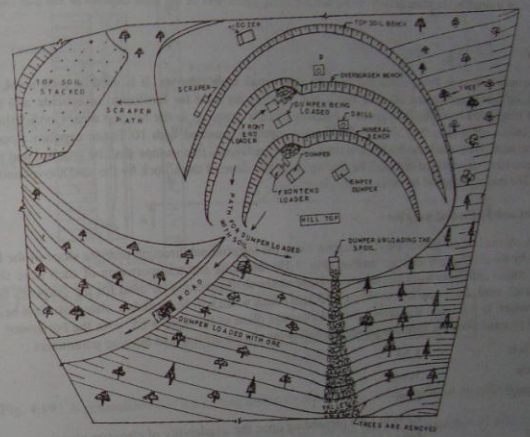


Fig. 4.18 : A generalized surface mining layout for hill top phosphorite deposit being worked by front end loader – dumper combination.

Production and Productivity

Production (given) = 1000 Tonne/day
Expected average OMS (3 shifts average) = 6 Tonne/man/shift
Total manpower requirement of the mine: 166.66 = 167 persons

Land Reclamation and Revegetation

As the winning process continues sufficient de-phosphorite area shall be created over which the scrapped top soil of at least 1m thickness shall be dumped and dozed properly by the bulldozer at an inclination of 1 in 20 towards the slope of the hill. When the total phosphorite winning process is over, the stacked top soil shall be sprayed (at least 1m thickness) on the remaining de-phosphorite area and also over the dumped overburden (limestone and marl) thrown on the valley of the hillock. The graded top soil is thereafter restored by amelioration and bio-fertilization by the agents for generation of micro biota. After bio-reclamation the reclaimed land shall be developed into a pasture land and afforestation shall be carried out by plantation of various kinds of plants (as already mentioned in the previous layout problem). The flower and fruit trees shall also be planted to make it a very good picnic spot. Over the steep slope area grasses of various varieties like cainas caian, cymbopogen citrates, bromas inermis, chloris gayana, etc. may be grown. Parks, shopping centre, guest house facilities, hotels, clubs, small zoo, etc. may also be constructed over the top of the hillock for making the hillock a very beautiful visiting place of interest.

Mine Drainage and Pumping : Consult layout problem No.8

Lighting, Fencing, Supply of Electricity : Consult layout problem No.2.

LAYOUT PROBLEM 16

Give a layout of a small scale slate quarry in the hill top area. Thickness of the slate deposit is 3m and overburden (top soil) is 1m.

SOLUTION

Slate now a days is being extracted in India from small scale quarries. It is used as rock-wool, tiles, bricks, damp course, surfacing, aggregates etc. and also a few amounts are used for fertilizers, insecticide and in the plastic industry. It is considered here that the approach roadway already exists up to the hill top area. Considering that it is a small quarry only a small capacity shovel (1.5m^3 bucket capacity) matched with 10 Tonne capacity tipper trucks are deployed in the mine. First of all the bushes and trees shall be removed , tree stumps shall be grubbed and thereafter top soil shall be excavated ,transported and shall be thrown along the slope of the hillock by the said shovel and tipper truck combination. The mining site shall be made totally clear.

Mining Layout and Method of working

A box cut shall be made over the slate deposit along the longitudinal direction near the slope of the hill. Box cut shall be made by the drilling and blasting method (Fig. 4.19). Mucking and transportation of such slate shall be done with the same shovel and tipper truck combination. These slates shall generally be used for the purposes of aggregate, brick making ,rock-wool, damp course ,etc. When sufficient bench width will be available ,slate slab shall be extracted in a systematic manner. In the figure it is shown that the cleavage plane of the slate is 65^0 with the horizontal. Blast holes shall be drilled over and along the cleavage plane. The following are the drilling and blasting parameters :

Depth of blast hole	:	3.2m
Dia. of blast-hole	:	51mm
Blast-hole spacing	:	1.5m
Burden	:	1.2 m to 1.8m (depending upon the availability of cleavage plane)
Charge per hole	:	560gm to 800 gm cartridge explosive with deck charging.
Explosive used	:	Superdyne or Kelvex-220.
Number of rows per blast	:	3

Drilling shall be done either by a small capacity wagon drill or by jack hammer.

Pattern of blast holes	:	Multiple 'V' cut
Initiation	:	By both down-line and trunk links of Cordtex detonating fuse and 6 to 10 number delays shall be used.

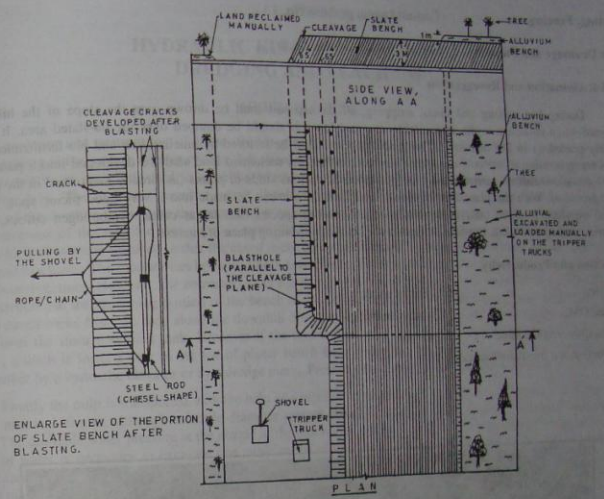


Fig. 4.19 : A generalized surface mining layout for slate mining of flat deposit at the hill top area

The purpose of blasting is to create well fractured lines for the purpose of extraction of well developed slate slabs.

After blasting is over chisel shape steel rod to be inserted into the fractured plane, rope/chain to be tied up on them and thereafter the slate slab shall be pulled out by the shovel as shown in the Fig.4.19 . The slate slab thereafter to be loaded over the tipper truck and shall be dumped in the polishing shop. In the polishing shop slates are sized and polished according to the market demand.

Main Equipment

The main equipment required to be procured are as following :

Main equipment	Total number
Shovel (1.5m^3 bucket capacity)	1
Tipper trucks (10tonne capacity)	5
Jack hammer drill	3
Wagon drill	1
Air compressor (7m^3/min capacity)	2
Diesel jeep	1
Centrifugal pump (50 l/s capacity, 35 kW ,440V)	1
Exploder, circuit testers, etc.	2
Polishing machine	1
Gang saw	1
Electric saw, etc.	1

Lighting, Fencing, etc. : Consult Layout problem No. 4

Mine Drainage and Pumping : Consult layout problem No. 8

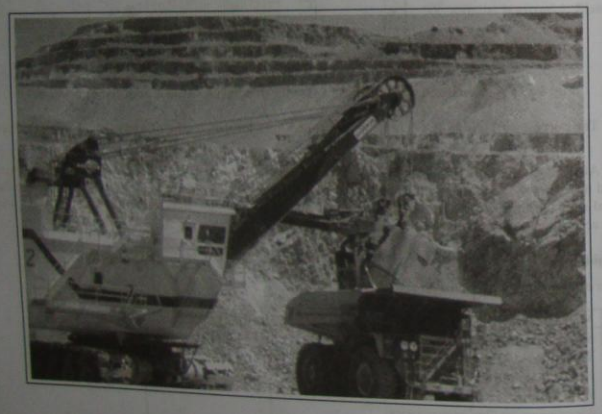
Land Reclamation and Revegetation

During box cutting and initial stripping, all the top soil shall be thrown over the slope of the hillock but afterward when sufficient width is available, the said top soil should be dumped over the de-slated area. It shall be properly graded, (1 in 20) manually. The graded top soil shall be restored by amelioration and bio-fertilization by the agents for generation of micro-biota. After bio-reclamation, the reclaimed land shall be developed into a pasture land and afforestation shall be carried out by the plantation of various kinds of plants (as already mentioned in the previous layout problem). With the plantation of flower and fruit trees it can be made into a very good picnic spot. The soil dumped on slope shall be cultivated with grass of varieties species like cainas caian, cymbopogen citrous, bromas intermis, chloris gayana, etc. to make the spot a very beautiful visiting place of interest.

Production and Productivity

Production	:	450 Tonne per day
Expected OMS	:	5 Tonne/man/shift
Total manpower in the mine	:	90 persons

(2 production and one maintenance shifts)



A metalliferous opencast mine under operation. A ropeshovel is loading on a dumper.

CHAPTER 5

HYDRAULIC KING, PLACER MINING DREDGING AND LEACHING

HYDRAULICKING

Loosening and transporting of loose, friable gold placers sands, loam, gravel, pebble, etc. can be done by hydraulic mining in the surface mining system. A hydraulic monitor is pointed towards the above mentioned ground and a high pressure water jet is delivered to loose the ground. The dislodged ground is thereafter transported by the water stream (Fig. 5.1). The hydraulic monitor mouth should be placed as close as to the working face to increase the efficiency of the monitor, to increase the amount of excavation of the ground and to decrease the consumption of water. The effectiveness of the monitor also depends upon the diameter of the nozzle and water pressure. But the monitor should be located at a safe distance so that the caved rocks from the bench do not cause any accident and damage to both the men and machines. The monitor can be moved both horizontally and vertically and can direct the high pressure water jet in various directions. However, the common practice is that first the monitor is pointed towards the toe of the bench to cut that portion and the remaining portion of the bench is allowed to cave down. The stream of water thereafter washes away the caved rocks from the face along the downhill direction by gravity action through the pipe line or channel and deposits over the storage settling tank. However if the surface terrain is such that the natural drainage of pulp is not possible , a ditch is formed near to the toe of placer bench from where the mixture of rock and water is transported upward either by a hydraulic elevator or by a dredge pump. Presently the dredge pump is most commonly used.

Finally the pulp is transported with the help of a pipeline or channel. The nozzle diameter of the monitor varies from 37 mm to 150 mm, although a nozzle diameter around 50 - 75 mm is the very good for mining of loose placer deposit. The desirable water pressure at the nozzle of the monitor is around 10 - 12 kg/cm^2 and an amount of (around) 10 - 40 m^3 of water is necessary to excavate one cubic meter of rock depending on the location of the monitor from the working face. In a properly selected hydraulic monitor, the effective range of jet is nearly half the head of the water. A water jet range around 50 m at a 100m water head is quite common. Water is stored in the tank above the working face to create a high head. More the head more advantages will be gained in the hydraulic mining system. In the hilly terrain ditches, tunnels, wooden or steel flumes are constructed to divert water into the water storage tank. Water is also pumped into the storage tank by pumps in the other cases but it is an expensive operation. From the storage tank water is supplied to the monitor with the help of steel pipes having optimum diameter and wall thickness. Air relief valves must

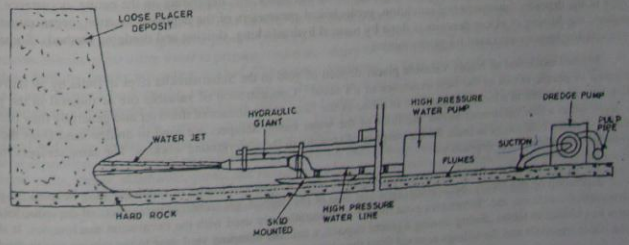


Fig. 5.1 : A Schematic Layout of a Hydraulic Mining System

be provided in all the vulnerable points in the pipe range. Channels are excavated or pipe lines are laid for transportation of excavated minerals or rocks, should have a gradient of at least 4° to 5°. In most of the cases wooden or steel flumes are laid to efficiently carry the materials along with water. The velocity of water stream to transport the excavated materials varies form 0.1 - 4 m/s depending upon the size of the materials (maximum around 300 mm) to be moved. Higher the

size of the materials higher would be the requirement of water stream velocity. Similarly the velocity of pulp through the pipe line also varies from 1.5 m/s to 4 m/s depending upon the diameter of the pipe line, size of the material to be transported and the presence of mixture of various ingredients. The mixture of water and rock transported is discharged in to the settling tank in the dumping place where the solid particles settle down and water is recirculated again into the water storage tank to reuse with the help of a pump having capacity ranging from 250 m^3/h to 1500 m^3/h at a head ranging from 50m - 80 m. The volume of ground handled by the hydraulic mining system may be around 500 t to more than 8000 t per day with an average productivity per man per shift is around 20t - 50 t.

The basic requirements of the system are as follows :

- The deposits to be mined should be friable, loose, soft and capable to disintegrate by the high water jet pressure,
- Availability of high quantity of water,
- Availability of storage place at a higher altitude compare to the mining benches,
- Availability of sufficient natural gradient to convey the dislodged materials (a gradient more than 6° is very good),
- Availability of handy movable hydraulic monitor with controls,
- Availability of good access for channels, pipes,
- Availability of sufficient capacity dredging pumps, etc.

PLACER MINING

A placer deposit is a loose mineral deposit formed either by chemical weathering at the place of origin of ore or formed by the mechanical weathering of ore body by the stream of water. Sediments are thereafter migrated to a nearly or far off distance and deposited layer after layer underneath the water. The placer deposits are also formed by weathering of very lighter ore by the wind action which are migrated to some nearby place or they are also formed by erosion and migration of the ore body by the tidal wave in the sea beach or formed by weathering due to the water current over the pre-existing placers in the offshore.

Gold, tin, platinum, diamond, ruby, opal, ilmenite , chromites, a few radioactive minerals, etc., are found profoundly in the placer deposit. Placer deposits are most of the time covered by overburden like alluvium, slits, sands and other noncommercial marine deposits. This overburden is removed by dredging, hydraulicking, ground sluicing (Fig 5.2), scrapers, dragline, bucket chain excavator, bucket wheel excavator, etc. depending on the nature, occurrence, type, and size of the deposits, environmental condition, geotechnical parameters of the deposits, location, organization, cost economics, etc. Mining of placer deposits is done by manual hydraulicking, sluicing and dredging methods. Sometimes scraper and draglines are also used for placer mining.

Manual extraction of highly valuable placer deposit of gold in the Subarnarekha river in India by an individual or a group of people is one of the best examples in it's favor . Concentration of valuable ore or mineral in the placer deposit may be present in a localized patch or in a line in the floor of the stream of the river and generally they occurred at a depth about a few centimeter below the floor of the water basin. Scrapers are used to dredge sand from the river Damodar in many coal mines in India for the purpose of stowing in the underground working. The same scrapers may be used for mining other placer deposits too.

In the surface of the land small capacity dragline, shovel, front end loader or bulldozer may be used for placer mining depending on the nature, occurrence, type and size of the deposit, geotechnical parameters of deposits, location, environmental condition, etc. Sometimes even mobile concentrators are used with the excavation machineries. It should always be planned to locate ,during mining a placer deposit, a suitable dumping yard near to the mining site to discharge all the waste materials and the tailings rejected from the concentration plant.

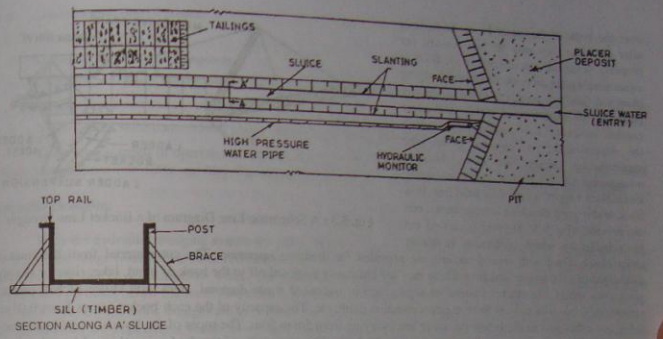


Fig. 5.2 : Schematic Layout of a Typical Placer Mining

Example

The Indian Rare Earth Ltd. is doing placer mining from decades and producing more than 4 lakhs Tonne of ilmenite and associate minerals like rutile, zircon, sillimanite, garnet, monazite, etc. Ilmenite is the bulk mineral produced from beach sands constituting 50% -60% of the output is further used for making titanium metal (5%) and titanium dioxide (95%). Titanium metal is mainly used in aircraft industry ,watches, submarines, missiles, ships etc. Titanium dioxide is mainly used as white pigment in the manufacture of paints and coating plastics and paper. Rutile is mainly used for making welding electrodes and as a titanium dioxide feedstock for production of titanium metal. Zircon is mostly used in ceramic glazes in tiles ,refractory, foundry, TV glass and abrasives. Garnet is mostly used as abrasive media and medium for sand blasting in the ship building industries as well as water jet cutting. Sillimanite is mostly used as refractory bricks lining in the high temperature furnace kilns and cupola of blast furnaces.

The beach sand mineral reserves are identified by exploration through extensive drilling and analysis of grade of Heavy Mineral (HM) content. The HM content in the in-situ deposit varies from 20%to 65% on weight basis with respect of raw sand, depending on the geo-morphology of the area. During mining, beneficiation of ore is done on site on a pre-concentrator plant using water to prepare sand-water slurry for beneficiation to increase the HM concentration level to about 90% by gravity separation method. The gangue consisting of light quartz/silica is removed from the slurry in spirals and dumped back to the excavated areas, leaving no void after excavation. The concentrate (90% HM) is transported to Concentrate Upgrading Plant (CUP) for further beneficiation to 99% HM consisting mostly of heavy minerals (viz. ilmenite, rutile, zircon monazite, etc). The CUP-concentrate is dried to bone dry condition in rotary fluidized bed driers in the Mineral Separation Plant (MSP) and fed to a series of separation equipment for individual mineral separation. The physical characteristics of individual minerals are used for separation. The heavy minerals viz. ilmenite, rutile, zircon, sillimanite, garnet and monazite are heavier than ordinary quartz/silica and have typical physical properties. The minerals behave differently during their physical beneficiation due to their difference in electrical conductivity, magnetic susceptibility, specific gravity, surface characteristics and grain size distribution.

DREDGING

The dredging is the process of under water mining of sand, loam, silt, clay and placer deposits of gold, tin, etc. lying over the floor of natural pond, lake, river, sea shore, etc. Sometimes artificial ponds, lake, water basins, etc. are also created to dredge the placer deposits in most economic way. The term "dredge" includes any floating vessel used for cutting, pumping or treatment purposes and other machinery to be used for or in connection with mining by means of dredging. The dredging is commonly practiced by the multibucket dredges and hydraulic dredges. In case of multibucket dredging placer deposits are cut horizontally layers after layers by the buckets up to the entyre thickness of the deposit by lowering, raising and adjusting the ladder of the bucket. The cut material is thereafter elevated and disposed off directly

over the bank of the water basin. Sometimes after elevating they may be first washed out properly, screened, concentrated in the floating concentrated plant accompanied in the dredging plant and finally disposed off in the processing plant for the extraction of metal. A dredger consists of several steel pontoons over which the superstructure is mounted. Over the superstructure bucket line drive, hoisting arrangement for the buckets ladder, swinging mechanism, hopper, winches for head line, bow lines, braking and clutching arrangements, etc. are provided (Fig 5.3). Digging is carried out by a ladder on which continuous revolving chain closely fitted with several buckets are provided for dredging operation. The cut material from the bucket is discharged into the hopper and from where they are ultimately disposed off to the bank of pond, lake, river or sea shore via various means like stacker system for segregation of oversized waste disposal, stem with sluice or belt conveyor system for disposal of tailings from the concentration plant, etc. The capacity of the each bucket varies form 0.05m^3 to 0.5m^3 and a dredger can dig below the water level varying from 5m to 50m. The ropes of the bow line and head line are anchored either to the bottom of water basin or to the bank. The movement of the dredge is achieved by releasing the ropes form bow line winch located in one side and tightening the ropes by bow line winches at the other side. The five line ropes are also used to stabilize the dredge.

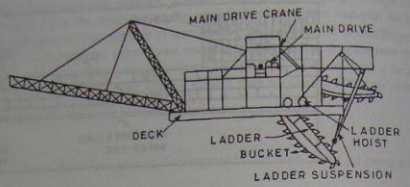


Fig. 5.3 : A Schematic Line Diagram of a Bucket Line Dredger.

The dredging is applicable under the following conditions :

- The placer deposit should be soft and contain either no or lass amount of boulders, timbers, etc.
- The placer deposit should be easily disintegrated and floor shall be smooth (for easy digging) having no fissures. Presence of fissures or cracks will result losses of water through them.
- Sufficient quantity of water must be present in the water basin, river, etc. so that a dredge can nicely be operated. Loss of water due to disposal of pulp on the bank should be compensated. If the amount of water present in the pond or lake is less , water is to be added to raise the water level. Amount of water requirement also depend upon the characteristics of placer deposit, presence of clay-type hard digging materials, digging depth, etc.
- The inclination of surface terrain should not be more than 1^0 to maintain efficiently water level in the water basin. It will also facilitate artificial construction of water basin.
- Reserve of placer deposit must be high enough to win the same for considerable years. During dredging a high mineral content smaller selected patches shall be avoided. Dredging should be carried out fairly all around the placer deposit maintaining an average grade of ore.
- Depth of the placer deposit form the water level is very important. It is the main criteria for selecting a dredger.
- The nature and amount of overburden also dictate the dredging method. The clay, silt, etc present over the placer deposit (termed as overburden) must be removed prior to excavate the placer deposit. Hence cost economic analysis is very essential to assess the economic viability to extract the placer deposit.
- The dredging area better be above the high flood level zone. Due care is to be taken to dig the placer deposit when water basins become totally full during the rainy season. Digging depth will be increased and there may be the problem of anchoring arrangement during that period.
- The dredging area in the very cold climatic region should be avoided because of frost and ice accumulated over the water will hinder the dredging operation. Costly thawing operation will be necessary prior to dredging operation to melt ice or frost.

Bucket Dredging

Bucket dredging can effectively be used even in low mineralized placer deposits. In the offshore special precaution is to be taken against the wind, typhoon, unfavorable weather and tidal condition, low centre of gravity of the dredge, etc. The following are the main advantages of bucket line dredging system :

- Low consumption of power,
- Lower operating cost,
- Lower requirement of manpower,
- Thin placer deposits can be winned easily with high control and efficiency,
- Smooth and clean surface at the bottom of the cut basin,
- It can dredge semi-consolidated placer deposit compare to a hydraulic dredge.

The average speed of operation is around 20 to 25 buckets per minute with an average production rate around 1000 m^3/day to more than 10000 m^3 /day. The average productivity is around 50 - 80 m^3 of material per man per shift.

Hydraulic Dredging

With the hydraulic dredging system the placer deposit lying below the water is extracted by suction pumps. One end of the pipe range (diameter around 150 mm to more than 900 mm) is connected to the pump whereas the other end is placed over the placer deposit below water. The open end of the pipe range has a screen where the larger particles are screened and particles below the mesh size extracted continuously in the pulp form by the pump and discharged over the dumping yard. The hydraulic dredges are very much useful for dredging silt, sand, gravel, etc. The pump units and other main accessories are kept over a vessel which moves from place to place. The digging depth is around 10m to 50m and the power of the pump may be 100 kW to more than 7000 kW depending upon the lift (depth of digging) and diameter of the pipe range. Floor of the placer deposit are sometimes agitated by the hydraulic jet. Hydraulic dredges are suitable for a particular selective placer deposit since it is unable to extract larger particles, unable to clean completely loose placer deposit, cannot extract sticky material, etc. The average percentage of solid by weight in the slurry is only around 15% to 18%. Fig. 5.4 Shows a schematic line diagram of a hole mounted dredging excavator under operation.

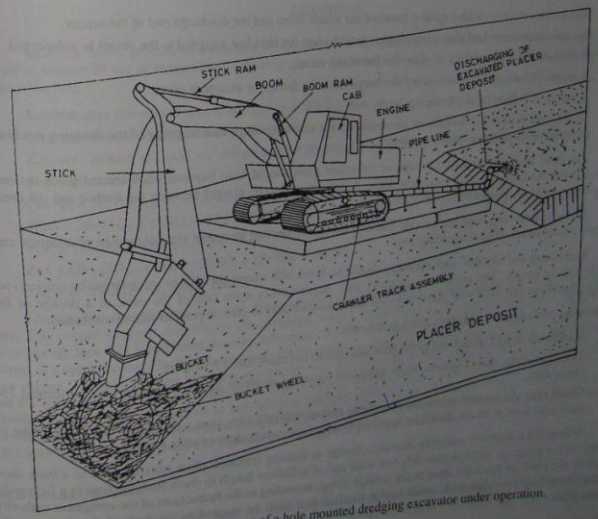


Fig. 5.4 : A schematic line diagram of a hole mounted dredging excavator under operation.

Some Safety Measures as Specified by the ILO

A dredge, floating treatment plant or other vessel should not be used in mining unless it has received the authorization of the competent authority. The application for authorization or use by the competent authority should be accompanied by drawings and specifications of the dredge, floating treatment plant or other vessel, buoyancy calculations in respect of the vessel made by a competent person and the results of all buoyancy tests carried out in respect of the vessel. The competent authority should not assign to any vessel a load line such that the vessel would thereby have a freeboard of less than 150 mm.

The hull of a dredge or other vessel used for mining should be kept sound and watertight. The interior of the hull compartments should be kept clean and, with the exception of those compartments in which the storage of ballast has received the approval of the competent authority, free of water as far as is reasonably practicable.

A dredge or floating treatment plant should be provided with four lifebuoys, two of which should be located in the bow and two towards the stern; suitable lifelines fitted around the vessel at a convenient height; a boat containing a lifeline not less than 15 m in length and a boat-hook , equipment ready for use with oars and rowlocks or other effective means of propulsion; an adequate number of life-jackets and notices so displayed as to indicate clearly the location of all life-saving appliances.

The life jackets provided should be worn by all persons when on board or when there is a danger of drowning. All life-saving appliances should be maintained in good order and condition and should be kept in a readily accessible location.

All employees engaged on or in connection with dredgers should be given training and instruction in life-saving techniques.

All dredges should be fitted with the following safety devices:

1. An electric return signaling system between the winch room and the discharge end of the screen;
2. An automatic alarm bell that is designed to sound when the load line assigned to the vessel is submerged.
3. A pendulum or other device to show the list of the vessel;
4. A means of indicating an obstructed discharge from the dredge or treatment plant;
5. A means of indicating the dredging depth; and
6. In the case of a bucket dredge, an automatic device to ensure immediate stopping of the dredging machinery in the event of an overload of the dredge.

The head and side lines of a dredge or treatment plant should have free and unobstructed place between the anchors and the dredge or floating treatment plant. A deflecting sheave, if used between the anchor and the dredge or floating treatment plant, should be securely anchored.

Every anchor for a head line or side line of a dredge or floating treatment plant should be of adequate capacity to hold the vessel under all working conditions

Hatchways and all other deck openings should be fitted with watertight seals or safeguarded by comings not less than 400 mm in height. The competent authority may require every open side of the deck of a dredge or floating treatment plant to be provided with guard-rails and stanchions suitable to prevent persons falling overboard.

The following provisions should apply to all guard-rails:

1. Stanchions should be secured to the deck and spaced not more than 2.5 m apart;
2. The guard-rails should consist of an upper rail located 1 m above the deck and a lower rail located 250 mm above deck;
3. Guard-rails should be constructed of suitable timber, scaffold tube, steel wire rope or chain and, in the case of steel wire rope or chain, should be kept taut by means of turn-buckles or other suitable devices.

Where the competent authority directs, a dredge or floating treatment plant working close to a bank should be provided with a gangway not less than 600 mm wide and of sufficient length to reach from the bow to a firm and stable position on the bank, or from the stern to the tailings dump, according to the instructions of the competent authority. The gangway should be provided with a substantial handrail and shall be secured to the deck of the dredge or floating treatment plant.

Every place where men are working should be adequately illuminated during the hours of darkness.

No unauthorized person should be allowed to work in the vicinity of the head or side lines whilst the dredge is operating nor permitted to interfere with any machinery, line or other appliance used in a dredging operation. The maximum number of persons that may be carried at one time in a dredge or other vessel used in connection with dredging or mining operations should be fixed by the competent authority The following safety requirements should be observed by all persons working on a dredge used for mining purposes:

1. Every member of a dredge crew when employed in outboard work on the dredge should use safety belts or life-jackets;
2. In the event of a man going overboard, the bucket line or cutter and suction equipment should be stopped immediately and the alarm given;
3. No person should step on any bucket, chain or revolving screen whilst it is in motion, nor ride on them.
4. The mine operator or manager should appoint a competent person to carry out specific inspections and controls daily before the commencement of operations,

All dredges and other floating installations should be provided with firefighting and emergency alarming facilities in a quantity to be agreed with the competent authority

On listening any emergency alarm, the winch man or other person in charge of the dredge or other vessel concerned should cease digging and take all necessary measures to correct the defect causing alarm and not recommence digging until the defect triggering the alarm has been effectively corrected. After the operation of the automatic overload stopping device, digging should not be permitted to recommence until the winch man has ascertained that the overload is cleared.

LEACHING

Leaching is the process of extraction of soluble metal compounds by the reaction either with acid or basic solvents or water with ore or waste dump. Leaching can be divided into two basic system viz-(i) Chemical leaching (ii) Bacterial leaching.

Sulphide ores like copper ore - Chalcopyrite ($CuFeS_2$)

Lead ore -Galena (PbS), lorn ore -Pyrite (FeS_2),

Zinc ore - Sphalerite (ZnS), etc and even silver are leached by the acidic ferric sulphate solutions.

The reaction are as following-

$CuFeS_2 + 2Fe_2(SO_4)_3 = CuSO_4 + 5FeSO_4 + 2S$

$2PbS + 2Fe_2(SO_4)_3 + 2H_2O + 3O_2 = 2PbSO_4 + 4FeSO_4 + 2H_2SO_4$

$FeS_2 + Fe_2(SO_4)_3 = 3FeSO_4 + 2S$

$ZnS + 4Fe_2(SO_4)_3 + 4H_2O = ZnSO_4 + 8FeSO_4 + 4H_2SO_4$

$2Ag + Fe_2(SO_4)_3 = Ag_2SO_4 + 2FeSO_4$

Chalcopyrite ore after reacting with ferric sulphate solution produces copper sulphate, ferrous sulphate and sulphur. This copper sulphate, dissolves in the circulated water and discharged onto a sump or storage tank. From the sump or storage tank it is precipitated on iron. Afterward copper is washed away into a settling tank from where it is reclaimed as a copper slurry. Finally it is blended with the copper concentrate and feed into the smelter for the extraction of pure copper. The solution used for leaching purposes may be regenerated after chemical reaction. Sulphuric acid of around 8% to 10% strength, sodium hydroxide or water may also be used for leaching copper ore. The sulphide ores are first broken into fragments and then air as well as ferric sulphate solutions are circulated alternatively to react with ore. Air supplies oxygen for reaction. After reaction is over ,the water containing metal salt is taken out through the tunnels or trenches built in the broken ore body and treated it accordingly as mentioned above.

Application and Operation

Since leaching process is very cheap it can successfully be applied to extract copper or other valuable minerals from the overburden dump dumped around the surface mining periphery or applied on low grade ores which are heaped

or in the mineral body located in-situ. Overburden dumps are generally very long, parallel and narrow in width. Depth of the overburden dump for each leaching operation is around 10m to 15 m. However if possible above the first leached overburden dump layer 2 to 3 folds overburden dumps may be placed one after another for leaching consecutively. It must carefully be noted that the overburden dump should be free from any vegetation, fines should be very less and if required fines from the top of the dump surface must be removed. Afterward the overburden dump is blasted or ripped for swelling enough (to create enough void space) so that leaching solution along with enough quantity of air (for oxygen) are circulated around the periphery of each particle of fragmented rock containing a little bit of sulphide minerals. Leaching solution are either trickled, injected or flooded. In trickled system a network of synthetic pipes are laid in such a manner that every square meter receives enough amount of solution (around 0.25 - 0.3m^3 /day) for about 10 to 15 days depending upon the thickness of the overburden dump. When the presence of fine in the overburden dump is more, the solution is injected into the dump with the help of a perforated pipe (mouth closed) and dipped into the bore holes made in the dump. When there will be no further soaking of leaching solution poured into the pipe, it will indicate the saturation limit of the strata. The copper bearing solutions are pumped into a tank to recover copper by iron cementation process.

Basic In-Situ Leaching Condition

The following minimum condition must be met before in-situ leaching :

- The ore should occur in a horizontal bed underlain and overlain by a relatively impermeable stratum.
- The host rock should have a good permeability.
- The ore must be located below the water table.
- The direction and velocity of regional water flow must be known.
- The ore should occur at favorable depth.
- The ore body should be of sufficient size and grade to justify the cost of production.

In-Situ Leaching Process for the Deposit Located below Ground

Under favorable geological and hydro-geological conditions, the process of in-situ leaching is carried out by injecting the suitable solvent into the ore body through injection holes. The solvent is allowed to pass /flow through the ore body for a certain period dissolving the desired metal content and the pregnant solution thereafter is recovered through production well by pumping.. The pumped out solution pass through normal hydro-metallurgical route in which metal is segregated. Generally wells are drilled on known geological and hydrological area over the ore body. A typical in-situ leaching operation normally has four different types of wells .

1. Injection well - Through this well, leachant is injected into the ore body at desired depth.
2. Production well – Through this, pregnant solution from underground is pumped out and sent to the mill for further recovery.
3. Monitor well - These wells are used to know the degree of contamination of ground water, if any.
4. Liquidated wells - In a typical in-situ leaching well field map, these wells are found distributed throughout the area. These are generally the exploration or other miscellaneous holes which were drilled to prove the ore body. but later on sealed . Sealing is must for aquifer isolation.

Normally, the well field design based on five spot or hexagonal pattern where the injection wells are located at the corners. Production wells are placed at the center of the pattern. The main objective the pattern design is to maximize the ore reserve addressed by each well within the operating spacing limitations. Factors like permeability and rate of leaching etc. are also considered in deciding the pattern.

After the completion of preliminary design, the delineation drilling / completion stage begins. All planned wells are of 150mm diameter and drilled little beyond the bottom of the mineralized zone. These holes are lined with 120mm diameter PVC pipes and the annulus between the PVC pipes and the holes are filled up with sand and gravel leaving the selected ore zone for leaching. Thereafter the holes are stimulated using a process called air lifting . Both injection well and recovery well are mutually reversible. In recovery well, a submersible pump (10 kW to 15kW) placed just above the ore zone to pump out leached out solution to the surface.

Monitor wells are installed in the periphery of the deposit about 300m from the outer line of operating wells to assist in 'excursion' detection in the of horizontal or vertical migration of the mining fluids. A rise in water level in wells

indicates a build-up of water pressure within the area requiring more withdrawal from the production well while a fall indicates too high a withdrawal rate. Hence a balance is always required.

Selection of solvent is done based on the type of mineralization and correlation of soluble and dissoluble minerals, type of minerals, associated mineral integration which determine the access of the reagent to minerals' surface.

Advantages

The method has the potential to work a deposit confined to unfavorable locality like incompetent host rocks, poor ground conditions or places having a large in-flows of water ,etc. This method is also suitable to work in the poor grade ore deposits. This method is also advantageous due to low gestation period , lower capital cost , high productivity, quicker return of capital , higher safety , very less prone to environmental damage and also require less manpower for operating the total system.

A bauxite opencast mine under operation. A backhoe is unloading bauxite on a dumper.

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CHAPTER 6

EXCAVATORS

Generally the selection of excavators depends mainly upon the following factors-

- Method of stripping and disposal of overburden (over casting or transportation of excavated waste);
- Diggability -This factor depends on the bulking factor, competency, abrasivity , strength, flow property of rock, fragmentation of rock, etc. The type and nature of rock like surface soil, weathered rock, soft and hard rock, etc. play important roles in digging characteristics of an excavator.
- Horizontal and vertical reach and reach below the grade;
- Operating condition and pit geometry,
- Maneuverability,
- Tonnage required (capacity) to be handled per unit time,
- Load bearing capacity of the floor rock,
- Power supplying system and arrangements,
- Blending requirement (use of number of small capacity machines are better in this case),
- Continuous operation (e. g. bucket wheel excavator, continuous surface miner, etc) or discrete operation (e. g. shovel, dragline, front end loader, rippers, scrapers etc),
- Surface topography,
- Use of the equipment for development purpose or excavation of ore or mineral,
- Nature, type and size of the materials to be handled (e. g. dusty, medium size, large blocky materials),
- Flexibility of operation,
- Requirement of assistance for loading by dumper, loader and transporting by other auxiliary machineries like dumper or conveyor ,etc.
- Indigenous availability of the machines and spare parts,
- Cost of the machine and availability of the capital,
- Ease of maintenance,
- Statistics and machine reliability,
- Operating and maintenance cost,
- Secondary use of the excavator,
- Climatic conditions affected by heat, cold, rain etc.,
- Overall safety.

EFFECTS OF SELECTING LARGER SIZE EQUIPMENT

Following are the merits and demerits of selecting larger size equipment-

- Productivity will increase,
- Operating and maintenance cost will decrease,
- Reduce the requirement of operating personnel,
- Pit geometry affected by the use of larger excavator, trucks and other larger auxiliary equipment. Larger trucks needs wider costly road (for two way traffic a wider road around 25m to 30 m with a clearance 2.5m to 3 m in between the two trucks crossing each other are necessary),
- Reduce requirement of number of machine units and hence there will be less congestion of machines.
- Larger machine requires efficient operating and maintenance facilities of equipment,

- A good wider, competent well constructed bench floor and haulage roads are essential to minimize the damage of tyres, crawler track assembly, floor penetration, etc. A good maintenance facilities for roads and bench floor are also essential,
- A proper survey is essential regarding the competency of ground floor through which the heavy earth moving equipment will move,
- Larger excavators needs larger capacity haulage units and larger auxiliary equipment to assist them,
- High capital investment is necessary to procure them.

FACTORS WHICH PLAY DURING PURCHASING OF EQUIPMENT

During purchasing any excavator or transporting equipment the parameters should be considered in great depth are -

1. Production and productivity, 2. Cycle performance, 3. Acceptance by the workman, 4. Standardization, 5. Climatic conditions like cold and hot weather, 6. Fire proofness, 7. Body configuration, 8. Visibility, 9. Operator's comfort condition, 10. Cost per unit of production, 11. Physical availability, 12. Life of the equipment and it's component, 13. Ease of maintenance, 14. Inspection and repair, 15. Requirement of maintenance personnel and their availability, 16. Workshop facilities and purchasing of special equipment for their maintenance, 17. After service facilities including spare parts by the supplier, 18. Capital cost of the equipment, 19. Operating and maintenance cost, 20. Erection and dismantling cost 21. Freight and insurance cost, 22. Availability and location, 23. Spare parts and technical service back-up; etc.

Selection of oversized machine should be always avoided, since it will cause over-investment. Optimum and best suited machine should be purchased after thorough techno- economic discussion with the manufacturer, investigation and evaluation in a similar conditioned surface mine where these type of machines are deployed for a long time.

SHOVEL

A shovel is a equipment which excavates the rock or ore by digging from it's operating base to upwards (stripping shovel in this case) and dump it either on an dumper (Fig 6.1) or railway wagon or over the spoil dump for backfilling after swing itself within its limit. It is a highly productive machine and capable to handle all types of ores, rocks ranging from fine to very hard blocky lumps, has lower operating cost, higher production and productivity etc. It requires lower power and has less wire rope cost. It also requires less manpower to operate and requires less surface preparation. It can also load in various mining conditions, has longer life, higher availability and can also do production by staying in the inclined terrain. However it has low maneuverability, low flexibility, effected by climatic and watery condition, bank slides, etc. Shovel can be used in strip mining method (loading, swinging and dumping of overburden material into the adjoining excavated area by overcasting), in tandem operations (where 2 or more shovels are used for rehandling of overburden successively), shovel pull back operation which is a combination of a shovel and dragline for handling the overburden.

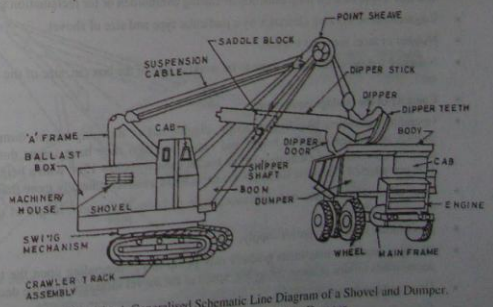


Fig. 6.1 : A Generalised Schematic Line Diagram of a Shovel and Dumper. Shovel is Loading on to the Dumper.

Shovel is used for digging a deeper cut, loading and overcastting and pull backing the spoil material, overburden removal in the contour mining in the hilly terrain, overburden removal in open pit mining system; excavation in the face and loading onto trucks, bunker or railway wagon, removal of top soil, construction of access roads and haul roads, opening up a mine by a box cut system, etc. A flat or mild gradient dry competent floor is a very good operating condition for a dipper shovel. The design of the shovel is such that only it's minimum mechanical parts (very nominal in weight, compare to the total weight) like the dipper stick, bucket etc. play important role in digging operation. The perfect motion of the dipper stick to reach and leave the face after loading facilitates will shorten cycle of operation. The bucket which is mounted with sharp teeth cut the rock or mineral body and break them with the help of pressure provided by the hoist and crowd actions. Efficiency of larger size shovels are much higher particularly during excavation of heavily blasted overburden rocks or coal. During operation assistance may be required either by a bulldozer or a front end loader to gather and muck piling of scattered rocks or ore. This will assist shovel to maneuver nicely near to the face and also effectively load them into the bucket.

Selection of Shovel

Selection of a shovel is basically done by considering the factors like -

- Requirement of daily production,
- Type and quality of the material to be excavated,
- Total coal or ore reserve, geotechnical parameters of mineral body or overburden rock. In the geotechnical parameter the thickness, compressive strength, tensile strength, shear strength, abrasiveness, dip, etc. of mineral body, coal and also overburden rocks are very important besides stratigraphy condition, ground stability, presence of contiguous seam, intrusion of hard bands, presence of fault, folds and other geological disturbances, ground water condition, etc.
- Bucket fill factor-larger shovel dig better than the smaller one.
- Swell factor,
- Working cycle time,
- Weight and maximum lump size of the materials to be excavated,
- Height of the bench and height of the cut (a shovel can dig material in a bench of height more than 55m),
- Whether the shovel is to be worked out in the overburden or mineral benches which are to be excavated directly,
- Whether it is to be used for production or casting overburden or for reclamation purposes,
- Requirement of working clearance by a particular type and size of shovel,
- Number of faces are to be worked,
- Nature of terrain, depth of the mine to be worked, size of the box cut, size of the area where shovel will operate, etc.
- Capacity of the haulage equipment,
- Operating parameters like digging height, digging radius, dumping height, dumping radius, etc. High capacity shovels have larger digging height and radius and also they have larger dumping radius, bucket size and capacity (generally for excavation of ore bucket capacity below 15m^3 with a height of bench ranging form 5m to 18m are selected. For overcasting bucket capacity of a shovel generally more than 15m^3).
- Over all dimensions-rear end clearance, angle of boom inclination, height of the excavator and height of the excavator cabin,
- Availability of electric power supply,
- Working gradient -maximum gradient may be up to 12° depending upon the type of shovels. Heavier shovel necessitates milder gradient (6° to 7°). Speed of the shovel increases with the decrease in gradient.
- Drainage condition,
- Ground pressure - due to the bucket filling with digging, swinging of the shovel and unloading of materials, average ground pressure may increase 15-20 times. The maximum ground pressure should not exceed the carrying capacity of the ground floor. Ground pressure should be within 1.5 kg/cm^2 for softer floor condition and 4 kg/cm^2 for other rock condition.

- Equipment availability particularly indigenously made,
- Climatic condition and drainage system,
- Reliability of the machine,
- Capital cost of the equipment,
- Operating and maintenance cost,
- Availability of the spare parts,
- Facility of after sales service,

During selection of a shovel the following points are also to be considered in depth-

- The power shovel is somewhat flexible in its operation,
- Since cost of the shovel is very high, it must be selected for a fairly long life project,
- Due to it's poor mobility, a power shovel is confined to operating nearly to a smaller area of coverage. So the selection of a single shovel for a widely spread areas of the mine is to be ruled out.
- Consideration must be given during planning of an adequate electrical power distribution system during selection of electrical shovel,
- Due to ease of operation of a shovel, its output does not hamper due to operator's fatigue,
- The power shovel has excellent digging capability due to its own weight, traction and high powered crowd and hoist motion. Out to crawler mounting, the power shovel has a distinct advantage over the tyred mounted loading equipment,
- Construction of shovel is rugged with the use of electric motors and sophisticated electric and hydraulic controls. The reliability and the efficiency of the shovel is exceptionally high,
- It can give high production,
- It can handle all types of materials, including large blocky materials.
- Operating conditions are fairly rigid,
- It may require supporting equipment for waste disposal.

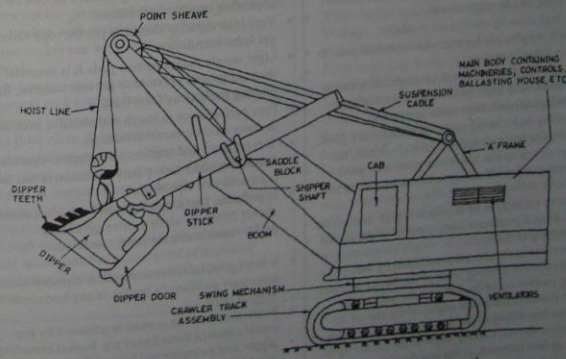


Fig. 6.2 : A Schematic Line Diagram of an Electric Rope Shovel

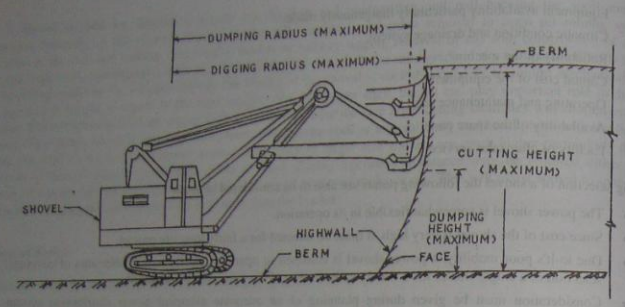


Fig. 6.2 (A)

Table 6.1 and 6.2 shows the comparison among rope versus hydraulic shovels and diesel and electric shovels respectively.

Table : 6.1 A comparison among rope shovels versus hydraulic shovels

Rope shovel (Fig. 6.2)	Hydraulic shovel (Fig.6.3)
• Higher initial cost, longer life, lower depreciation cost • Usually have a good resale value • Heavier basic machine weight, less mobile on site depending on site requirements. • Very high capacity shovels are available. • Heavier weight calls for wider heavier track pads to reduce ground pressure. • Very good performance even in tough and abrasive rock and also in dusty environmental condition. • It is bulky, requires more space to operate, • Reliability under heavy duty condition is very good. • In pollution free atmosphere and high altitude the shovel driven by electric drives gives very good performance but efficiency is less as compared to a hydraulic shovel. • Selective excavation (if required) is poor in this case since it has a poor stabilizing capacity of the bucket to a particular point. • Penetration in the ground is achieved by the combination of pushing force of the dipper arm and lifting motion of the bucket with the help of the boom (fixed to particular angle) of the shovel. It achieve less filling of the bucket as compared to a hydraulic shovel. Cutting angle of the bucket tip is more or less constant. It has less digging force.	• Lower initial cost, shorter life. • Resale value is usually poor • Lower basic machine weight, more mobile on site depending on site requirements. • Very high capacity shovel as per the rope shovel is yet to be manufactured. • Light weight makes narrow pads.It is essential to increase ground pressure to prevent machine form pushing itself away from bank. • Poor performance in tough and abrasive rock and also in dusty environmental condition, • It is compact and requires less space to operate. • Reliability under heavy duty condition is less as compared to a rope shovel, • In pollution free atmosphere and at high altitude the shovel driven by electric drive gives best performance under the same condition, • It has a good selective excavation capacity since it can nicely stabilize its bucket in any horizontal and vertical point, • Penetration into the ground is achieved by the combination of three forces e.g. bucket crowding, dipper arm swinging and boom lifting. It achieves good filling of the bucket. Cutting angle of the bucket tip can be changed. It's digging force is very high.

Table 6.1 continued

• Prior blasting is required for good diggability. Bucket fill factor is moderate. • It cannot be used as a backhoe. • All diesel and all electric shovels come under this category. • Operation is limited in the steep gradient. Dozer or wheel loaders are required to assist them for accumulating the muck piles. • Requirement of self movement power is more because of it's bulkiness and heavier in weight. • Utilization of power is poor to moderate. • Operation is jerky and cycle time is more. • Larger machine dimensions-front-end geometry, centre of gravity further from center of rotation, machine needs more counter weight. • Higher weight ratio to bucket capacity, contributes to absorption of shock loading, contributes to longer life. • High resistance to shock and fatigue due to high protection from cast steel construction and rope operation. • Boom remains at constant angle during digging and only limited assistance to penetration effort is necessary . • Bucket fixed to handle, tooth direction can not be changed during digging, bucket spends more time in bank to fill, more tooth wear. • End of boom represents maximum height to which bucket can be raised. • Bucket at maximum height has low digging power at tooth point. • Narrow bucket-gives good visibility and aids careful truck loading with widely varying lump size. • Narrow bucket has more swell effect on loaded material. • Due to bottom opening door in the standard equipment rock loading has less impact on truck bodies and suspensions. • Level floor clean up is limited. • Steering and movement is difficult. • Under identical condition noise and vibration level is higher as compared to a hydraulic shovel. • Efficiency of power utilization is lower under the identical condition. • Although maintenance cost is higher due to high priced spare parts but requirement of degree of maintenance is less as compared to a hydraulic shovel.	• Prior blasting may not require for softer and friable strata condition. Bucket fill factor is very good. • It can be used as a backhoe. • In this case power is generated either by a diesel engine or by a electric motor which supplies power to a number of hydraulic pumps via gearbox system for crowd, hoist, swing, travel, etc, mechanisms. • Steeper gradient may also be negotiated. Generally dozer or wheel loaders except extreme conditions are not employed to assist them for accumulating the muck piles. • Requirement of self movement power is less because of its less bulkiness and less weight compare to a rope shovel. • Utilization of power is highly efficient. • Operation is smooth and cycle time is less. • Shorter front-end geometry, center of gravity closer to base of machine, calls for less counter weight. • Lower weight ratio to bucket capacity, less resistance to shock loading, contributes to shorter working life. • Lower resistance to shock and fatigue mainly due to plate welded construction and non absorbing hydraulic operation. • Boom angle can change during digging, can assist with bucket penetration. • Bucket controlled by wristing cylinders allows tooth direction to be changed during digging, wrist action fills bucket with less time in bank, less tooth wear. • Arm connected to end of boom allows greater digging heights. • Bucket at maximum height still has high digging power at tooth point. • Wider bucket reduces operator visibility when rock loading. • Wider bucket has less swell effect on loaded material. • Bottom opening doors available usually at reduced capacity. Maximum capacity normally forward tip type which has adverse effect during discharging load into trucks. Extra cost is required to truck repairs and their working life become shorter. • Level crowd permits better floor clean up. • Steering and movement is less difficult. • Shovel operated by the electric drive has a very less noise and vibration effect. • Efficiency of power utilization is higher. • In this case the cost of maintenance is cheaper but requirement of degree of maintenance is higher as compared to a rope shovel.

Table 6.2 :A comparison among diesel and electric shovel

Diesel Shovel	Electric Shovel
• Applicable for doing a particular work for a shorter duration.	• Applicable for doing any work from shorter to longer duration.
• It is used where electric power is not available i.e. in remote location.	• It is used where electric power is available.
• It is very highly mobile equipment.	• It is less mobile equipment. It's mobility depends upon the shifting of switches, transformers, cables, etc. which are very cumbersome process.
• It handles comparatively smaller sizes hard and abrasive fragmented ore or overburden rock.	• It is capable to handle huge amount of blocky, hard and abrasive fragmented ore or overburden rock.
• A little bit complicated in construction having less reliability c as compared to an electric shovel.	• Very simple and robust in construction having very high reliability.
• Machine available time is moderate.	• Machine available time is very high provided there is no interruption of power.
• It has a good digging characteristics.	• It has moderate digging characteristics.
• All diesel rope shovel has a diesel engine which supply power to the crowd mechanism, hoist mechanism, travel mechanism, swing mechanism, etc. In hydraulic shovel power is generated by the diesel engine which supplies power to a number of hydraulic pumps via gearbox system for crowed, hoist, swing, travel etc. mechanism.	• All electric rope shovel has an a.c. induction motor, swing generator, hoist generator, crowd generator, exciter, etc. to operate swing mechanism, hoist mechanism, crowd mechanism, travel mechanism, etc. In hydraulic shovel power is generated by an electric motor which supplies power to a number of hydraulic pumps via gearbox system for crowd, hoist, swing, travel, etc. mechanism.
• It has high operating and maintenance cost and shorter life as compared to an electric shovel.	• It has low operating and maintenance cost and longer life.
• All diesel shovel has less digging capability below the floor as compared to an electric hydraulic shovel. All diesel shovel also does not have any 'wrist action' and hence more problem may be faced during digging unblasted toes. Noise and vibration level is more.	• All electric shovel has less digging capability below the floor compare to an electric hydraulic shovel. All electric shovel also does not have any 'wrist action' and hence more problems may be faced during digging unblasted toes. Vibration and noise level is less.
• Cycle time is more.	• Cycle time. is less as compared to a diesel shovel.
• Utilization of power is poor.	• Utilization of power is moderate.

OPERATING PARAMETERS

(i) Dumping (discharge) height (maximum)-At a particular boom angle it is the vertical distance between the level (where a shovel rests) and the bottom most point of the bucket under tipping condition while the boom and all other accessories attached on boom are at their full extended position under that condition (Fig 6.2(A)). Higher the boom angle higher would be the dumping height for a particular shovel. A dumping height of around 46 m is common nowadays.

(ii) Cutting (digging) height (maximum)-At a particular boom angle it is the vertical distance between the level (where a shovel rests) and the top most point of the bucket (tip of the teeth) when the boom and all other accessories attached on boom are at their full extended position under that condition (Fig 6.2(A)). Higher the boom angle higher would be the cutting height for a particular shovel. A cutting height of 51m or more is not uncommon today.

(iii) Dumping (discharge) radius (maximum)-At a particular boom angle it is the horizontal distance between the vertical swing axis of the main body of the shovel and the vertical centre line of the bucket when the boom and all other accessories attached on boom are at their full extended position under that condition (Fig 6.2(A)). Higher the boom angle lower would be the dumping radius of a particular shovel. A dumping radius of 65m or more is possible at present.

(iv) Maximum digging radius-At a particular boom angle it is the maximum horizontal distance between the tip of the bucket (tip of the teeth) and the vertical swing axis of the shovel (Fig 6.1(A)). Higher the boom angle lower would be the digging radius. A digging radius of 10m or more is quite common at present.

(v) Bucket fill factor = $\frac{Actual\ volume\ of\ the\ material\ inside\ the\ bucket}{Volume\ of\ the\ bucket} x100$

It depends upon the degree of fragmentation of mineral body, rocks or physical characteristics of the rock or ore i.e. the size and shape of ore or overburden rock, stickiness, watery condition, diggability, soft ,loose and fragile characteristics, and also on the size of the bucket.

Bucket fill factor for the following materials are as follows:

Material	Bucket fill factor
• For easily digging rocks like sand, loam, clay etc.	0.75 to 1.0
• Average digging rock like well blasted rocks	0.55 to 0.75
• Difficult digging rocks like bulky, irregular shaped or rugged rock which is very difficult to scoop up	0.35 to 0.55

(vi) Swell factor = $\frac{Weight\ per\ unit\ volume\ of\ solid\ rock\ mass\ in\ the\ bench}{Weight\ per\ unit\ volume\ of\ loose\ rock\ mass\ after\ blasting\ or\ loosening}$

It indicate the amount of increase in volume of a rock mass in percentage after expanding due to loosening by digging action or by blasting. Poor fragmentation due to blasting will increase the amount of swell since heavy amount of voids will be present in between the boulders.

(vii) Bucket Factor = Fillability x (Swell Factor) $^{-1}$

Fillability-Because of the slope line in front of the bucket the struck capacity of the bucket is reduced to some extent depending upon the size of bucket and the design. It is determined by the field observation and experience. The fill ability of the bucket is expressed in percentage. It is always less than 1. Reciprocal of the swell factor is taken into account for calculating the bucket factor.

(viii) Swing Factor-Standard cycle time of a shovel is based on it's 90° swing for loading. This cycle time will increase or decrease depending upon the increasing and decreasing angle of swing (a shovel can generally rotate at 4 to 5 RPM). So in such a case a certain factor is to be multiplied to correct the shovel cycle time. This factor is known as the swing factor for which a standard chart is provided by the manufacturer. This swing factor is 1 at 90° and it increases or decreases depending upon the increasing or decreasing the swing angle respectively corresponding to that standard 90^0 angle of swing.

(ix) Cycle Time-It is the total time taken by a shovel to complete one full cycle of operation starting from the crowding operation into the face to swinging, dumping and again coming back to the face for crowding operation.

Total cycle time = Crowding (digging, loading, hoisting) time + swinging time towards the truck + dumping time on the truck + swinging back for crowding operation.

In the shovel operation some amount of delays are always present which include the followings-

- Moving the equipment of proper positioning,
- Delay due to non availability of the haulage unit,
- Floor cleaning by the bulldozer for the movement of the shovel,
- Dressing overhanging ledges in the high wall and also blending operation for quality control,
- Improper spotting of the haulage unit and hence poor unloading,
- Delay due to fatigue of the operator, unskilled operator, etc.
- Due to poor blasting of rock when big boulders are formed,

- Adjustment of the boom angle and other mechanical parts.
- Power cable shifting and damage of machine breakdown-both the electrical and mechanical units.

Time also lost due to rain, fog, tremendous hot waves, cyclone, repair, maintenance and overhauling, shift changing, interruption of power, etc.

(x) Effective overall percentage of utilization of a shovel = $\frac{Actual\ working\ hours}{Total\ hours} \times 100$

(xi) Availability factor = $\frac{Schedule.shift.hours - Mainenance.hours - Breakdown.hours}{Schedule.shift.hours}$

(xii) Utilization factor = $\frac{Schedule.shift.hours - Mainenance.hours - Breakdown.hours - Idle.hours}{Schedule.shift.hours}$

(xiii) During calculation the capacity of a shovel the following points are also worth to mention-

a) Spotting time of dumper-If a skilled dumper operator and spotters are employed then the dumper spotting time will be around 20 to 30 second. Spotting can be done in one side or on both side of the shovel. In case of the double spotting the spotting time tends or becomes equals to zero.
Loading time-it is the time taken by a shovel to load a truck or a railway wagon.

b) Total loading time =Number of passes required by a shovel to fill a truck x Cycle time by the shovel.

Number of pass = $\frac{Tonnage\ rating\ of\ the\ truck}{Tonnage\ per\ pass\ by\ the\ shovel}$

Tonnage per pass by the shovel =Capacity of the bucket x Bucket fill factor x Bucket factor x Bank density.

c) Dumping (discharge) time- depending upon the operational condition dumping time may vary from 50 seconds to around 2 minutes.

d) Cycle time of a dumper =Spotting time + Loading time + Travel time with load + Spotting time to discharge + Discharge time (either in the stockpile or in the bunker or in the crusher plant) + Travel time with no load + Waiting time for loading (this waiting time may be reduced to zero if number of dumper required per shovel is perfectly matched).

Broadly shovels are divided into two categories which are as follows -

(a) Rope Shovel (b) Hydraulic Shovel. .

Both the diesel engine and electric motors are used as prime movers for both the shovels although all diesel and all electrical shovels are fall in the category of "Rope Shovel".

Rope Shovel (Dipper Shovel) (Fig 6.2)

It contains the following main units-

(a) All diesel operated rope shovel contains the following main units-diesel engine as prime mover, crowd mechanism, hoist mechanism, travel mechanism, swing mechanism, front attachments, undercarriage unit, control units, etc.

(b) All electric motor operated rope shovel contains the following main units-motor generator set (containing an a.c. induction motor, swing generator, crowd generator, hoist generator, etc) crowd mechanism, hoist mechanism, travel mechanism, swing mechanism, undercarriage unit, pneumatic control system for various braking operations etc.

In the rope shovel a bucket is rigidly attached to the end of the dipper stick. The dipper stick can move forward or backward. It is supported by a pinion and is held by a cable to the bucket side. The pinion and rack on the underside of the dipper stick is in constant mesh which are attached in proper position on a heavy structural boom. The lower end of the boom is pivoted with a turntable, while the top end is held in position by means of a boom hoist cable. The superstructure or the turntable carry the operators control room, mechanism for hoisting, crowding, swinging and traveling etc. of the shovel. The crawler mechanism supports the whole shovel unit and also provides the traveling and steering motion of the shovel. Shovels may be rope type or non rope type and bucket size varies from 0.3 m^3 to 200 m^3.

It is already mentioned that a rope shovel incapable to do clean up works. Hence a bulldozer is necessary to assist it. Compare to a front end loader a shovel has a very high breakout force for crowding action. It can dig smoothly down the grade to about 50% of the length of the dipper stick and upto about 65% of its boom height in case of rope shovel. Whereas in case of hydraulic shovel the below grade digging is about the full length of the dipper stick and up grade digging is about the full length of the boom height. A shovel can handle coarser materials safely in the face and unload them over the dumper more efficiently by swinging. Hence it requires less space for its operation. It has a very good life span (around 10 to 20 years) and very good resale value. But a shovel is a very expensive equipment and requires higher priced backup machinery compare to a front end loader. It has higher operating cost compare to a front end loader. It is also less mobile.

Front Attachments of a Dipper Shovel (Fig. - 6.4)

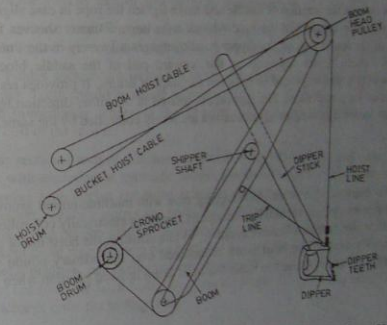


Fig. 6.4 : A Schematic Line Diagram of Front Attachment of a Rope Shovel

The front attachments of a dipper shovel consist of a bucket (mostly bottom discharge type), a dipper shaft (in 2 piece member), a boom (in 2 piece member), various sheaves, pulleys, hoist line, boom line, retract line, crowd line, trip line, etc. The massive steel boom bottom end is connected with the front deck of the shovel and the top end is provided with two sheaves for hoist line and pulleys for the 4-part boom line. In the centre slot of the 2 piece boom member a saddle block and shipper drum are pivoted as the shipper shaft. The saddle block (which is the dipper stick driving mechanism) and the shipper shaft provide positive crowding or retracting action with the help of rack and pinion arrangement. They also provide better control, less failure and reduce the maintenance cost. Hoisting of the boom is governed by the four part boom line. One end of the rope is anchored with the hoist drum and passes around the vertical pulley attached with the 'A' frame which is mounted at the other end of the shovel body. The rope passing from the pulley of the 'A' frame passes round the vertical pulley attached to the left hand side of the frame passing via. round the right hand side pulley attached at the tip of the boom and horizontal pulley attached with the frame in the left hand side and finally it is anchored with the 'A' frame. A brake and cone clutch control the rotation of the boom hoist drum. By the gravity action the boom is lowered down with the strict control of the brake mechanism. But during hoisting the boom culch is engaged with the hoist drum mechanism and boom is lifted up at a desire angle. In bucket hoist mechanism

hoist cable passes round the right hand side vertical hoist sheave and finally anchored with the boom passing via pulley attached with the bucket and left hand side hoist sheave.

The boom is made in 2 sections. The lower section is single and welded construction having 2 side girders and 2 cross girders with cast steel boom feet and shipper - shaft bearings welded into the structure. The end of the lower boom section is tapered and pivoted on the shipper shaft bearing structure. The lower end of this unit is connected with the centre of the revolving frame. The upper boom section is constructed by 2 fabricated H-beams and reinforced by suitable cross members. A cast steel boom point sheave is attached at its upper portion. At its lower end there is provision for a hinge connection to an extension of shipper - shaft bearings whereas at its upper end there are the attachments for the bridge strand suspension arrangement of the total boom. The dipper handle is a heavy structural steel member lower end of which carries the bucket. At the bottom of the handle a key is provided which engages the keyways in the saddle-block liners through which connection is made to the handle-stabilizing device. For equalizing the crowd lines a handle stop arrangement is mounted on a large vertical pin. The standard type buckets are of welded construction. The door of the bucket is made of alloy steel casting with the hinges and the latch-bar guides are integrally casted. The latch-bar keeper is the cast integral with the dipper front. The tripping of dipper is controlled by a magnetic valve operated trip mechanism. The saddle block accommodates an air-actuated type cylinder which operates the dipper latch by means of a rope. It is operated by a thumb latch switch placed on the crowd master controller. Presently the flood lighting system not only mounted in front of the machine, but also they are provided over the boom.

The large diameter shipper shaft mounted over the lower boom structure carries the saddle block. The alloy steel casted saddle block which has bronze-bushed hubs and renewable liners, guides the dipper handle. A device is fitted with the saddle block which retards the rotation of handle and make tighten the rope in casc slight slackness developed in the hoist rope during the crowding action. On the saddle-block hubs large diameter sheaves for the crowd and retract cables are mounted. A key at the lower part of the dipper handle engages a keyway in the liners and these rotate along with the movement of the handle. Liner segments at the forward end of the saddle block have the rack-a-track arrangement which engages with a pinion placed in between the shipper drums. It provides resistance to the rotation of handle through a spring loaded friction brakes. The sprocket is rotated in the either direction by the crowd-retract chain drive by a boom foot drum and crowd as well as retract cables are nicely controlled by the same.

Hoisting Unit

The hoist motor is connected to a drum (split lagging type with machine-turned groove) via the gearbox unit. The motor is connected to the primary pinion shaft by a smoothly engaging electrically controlled, air-actuated clutch. In small capacity shovel the clutch also acts as a flexible coupling. While hoisting, the hoist motor acts as a motor but while lowering it acts as a regenerative brake. The hoist brake is mounted over the primary pinion shaft is of spring-set air released and electrically controlled and it is used to make the hoist drum in fixed position when desired and shall be 'on' position as soon as the power is cut off.

Crowding Unit

In a smaller capacity shovel, it consists of a cable drum connected to a motor via. the gearbox- unit. It, is located at the forward end of the revolving frame casting. Rotation of the cable drum in one direction causes the handle and dipper out and rotation in the opposite direction causes handle to retract. Crowd ropes are wedged-anchored at the rear of the handle on the stop casting via. a half-sheave segment positioned by a large diameter screw for the purpose of adjustment of crowd and retract ropes. The loop of the crowd rope passes over a half - sheave at the middle of the crowd drum under the drum gear rim. The motor is provided with spring set, air released type automatic brake and air-actuated overload slipping clutch.

Both the crowd and retract mechanism is governed either by chain or cable. In cable type system, the rope attached with the foot drum (it has two parts), passes round the shipper drum and pulley attached at the rear end of the dipper stick wrapped and anchored to the left hand part of the foot drum after passing via the shipper drum. Similarly the retract line takes off from the left hand side drum and anchored to the front side of the dipper stick after passing via the shipper drum. During rotation of foot drum in the forward direction, the crowding rope coils in the drum, causing dipper stick to move in the forward downhill direction for digging (crowding) and pull down the retract rope which will be uncoiled to balance the system. During reverse rotation, the crowding rope will be released whereas the retract rope will be coiled in the foot drum; pulling the dipper stick up (retract).

In the chain type of crowding mechanism, shipper shaft accommodates a pinion (sprocket wheel) which intermeshes with the rack embedded on the dipper stick. The rotation of the pinion either in the clockwise or anticlockwise causes the crowding and retracting action of the dipper stick.

Ballast Housing

The ballast housing is provided at the rear end of the machine body and the capacity of such housing depends upon the capacity of the shovel. Generally pig iron or steel scraps are used as the ballasting materials.

Bucket and Teeth

For high capacity shovel the buckets are casted into one piece by manganese steel, open at the top and close at the bottom by a hinged type door. The smaller capacity shovel buckets are generally welded steel box structure, which are lighter in weight. The lip of the bucket is provided with replaceable four to six number manganese steel teeth to negotiate maximum abrasion during digging. The life of a bucket teeth are only 60 to 80 hours in hard sand stone but after doing some preventive measures their life may be improved. The preventive measures are as following-

- When one side of it is heavily worn out, it should immediately be opened and refitted in reverse position.
- If they are heavily worn out, then they may be rebuild to their original position by first welding with steel followed by welding with high tensile manganese steel on the top surface. This welding method will again increase the life of the teeth by another 50% of the total original life.
- In case a highly abrasive rock is to be encountered , the top surface of the teeth may be welded with high abrasive resistant tungsten-chromium-cobalt alloys. It will increase the life of a bucket teeth in many fold.

Generally tooth holder sockets are welded with the lip of the bucket where teeth's shanks are inserted and fastened by a removable pin. Teeth should be strong enough to withstand the entyre forces produced by the shovel. Dull teeth should be removed immediately otherwise it will unnecessary waste fuel energy, reduce diggability and also increase the maintenance cost of the shovel.

The bottom cross sectional area of a bottom discharge type bucket is large as compared to the cross sectional area at the top. Presently with evolution of hydraulic shovel, the bottom discharge buckets are steadily replaced by the side discharge or overturning type bucket for doing selective mining and also for providing the advantage of 'Wrist Action'. In most of the bucket, welding matrix are provided both inside and outside of the bucket. In case in any part excessive wear is visualized, that portion may be reinforced by welding manganese steel strip. The door of the bottom discharge type bucket is kept close with the help of a latch bar into the latch holder by the gravity action while the teeth of the bucket is in horizontal position. But during unloading, when the teeth of the bucket is kept in the vertical position and the latch bar is pulled out from the latch holder by a trip cable, the door opens by it's own heavy weight as well as load on it and material discharge onto the haulage unit.

Shovel Rope

During shovel operation ropes of the rope shovel are subjected to severe shock loads on it due to heavy load, various to and fro motion, etc. They are also subjected to bending stresses by sharp bending on pullies, wear due to friction with rock and also corrosion fatigue due to exposure to the atmosphere. Beside these hitting by external objects, manufacturing defects, bad quality, etc. impair the life of a rope. A time comes when a rope breaks, which may causes severe accidents to both man and machine and a huge amount of time will be lost to replace the cable. A wire rope which on inspection shows the sign of thinning of the top surface wires or their breakage and show a sign of expiry of life, should be changed immediately during maintenance.

Generally the life of a good rope is many hundreds of hours. Use of torque converter will increase the life of a rope to some extent.

Prime Movers

The power is supplied to the shovel either by diesel engine or by a. c. electric motor. For smaller capacity shovel generally one diesel engine or one electric motor is provided. The power is transmitted from them to all the crowding, hoisting, swinging and propelling mechanism via fluid coupling gearboxes, clutch system etc. 'V' type diesel

engines are generally used with improve fuel system to control the speed of the engine depending upon the requirements during crowding, hoisting, swinging, traction, etc. In high horse power engine for enhancing maximum power efficiency and also to work at high altitude 'Super Charger' or 'Turbo Charger' are generally provided.

A 'Supercharger' is simply a blower type device which forces air at high pressure into the combustion cylinder. This way it supplies enough quantity of oxygen which present in the air into the cylinder for burning more amount of fuel to enhance more power with the same engine. Because of the suction stroke by the piston in the cylinder and also pumping action by the supercharger the atmospheric air will flow inside the cylinder at high velocity and will be pushed further by the atmospheric air itself.

A 'Turbocharger' is a device which incorporates a set of turbine vanes and is driven by the high velocity exhaust discharge from the engine. Because of the rotation of the vanes at a very high speed (around 1400 to 1600 rpm) air forces into the engine at a high velocity. The exhaust gas channels through the nozzles for increasing the velocity of gas which thrust the vanes with high force for producing high R.P.M. The output of the individual cylinder are synchronized together for smoothening the action. The whole process will create a little bit back pressure but it's magnitude is far less as compared to the inlet pressure. In this way the waste energy of the exhaust gas is utilized to increase the efficiency of the combustion engine which will definitely increase the power. This exhaust gas also assist in increasing compression. One side of the turbocharger is continuously in touch with the hot exhaust gas which heated it up and the other side is in contact with the incoming air which cools down the same. So the engine is to be remain idle for a few minutes after doing a heavy work to cool down the turbocharger. It is lubricated by the lubricating oil supplied automatically by the crank case of the engine. Turbochargers are also incorporated in the diesel engine of various heavy earth moving machines.

The diesel engine cylinder is cooled down by the oil cooling nozzles. In high capacity shovel high voltage around 3.3 KV or 6.6 KV is supplied to an A. C. squirrel cage induction motor which drives several D. C. generator. The generator in turn supplies power to several motors to perform. the different operations like crowding, hoisting, swinging, propelling, etc. Most of the electric shovels are controlled by the Ward-Leonard control system. The electric shovels are cheaper than a diesel shovel, highly efficient, easy to control, maintain and repair, maintenance cost is less, easy and quick to start and stop and also highly reliable.

In a smaller capacity electric shovel an A.C. motor driven generator set is provided for supplying power to the high peak duty D.C. motors for swinging ,hoisting and crowding operations.

For excitation of the motor generator set, forced auxiliary ventilation of all the motors, etc. separate squirrel cage induction motors are provided. Auto transformer starting is used in the motor-generator set. Regenerative type braking system is used to all the crowd, swing and hoist motors. An auxiliary transformer is also provided in the machine for supplying medium voltage to the auxiliary motors for exciter, generator, air compressor, auxiliary fan and low-voltage supply for lighting system. The air compressor supplies air to operate various clutches and brakes. The collector rings are generally provided below the revolving frame and takes current from heavy duty shoes located on the base. The collector rings are well designed, properly insulated and have sufficient capacity. The cable entry box is of a very high capacity having bell-mouth gland. The master controller for controlling the speed and direction of rotation of each motor is placed at the operator's cab.

Under Carriage Unit

It consist mainly the main body, turntable and crawler track assembly. It supports the whole machine weight and assists traveling of the machine, controlling, steering, stabilizing the machine with out-rigger unit (if present) as well as by ballast chamber (ballasts are kept into it), assists erection and repair work with the help of crane gantry (optional) attached on the revolving deck, etc.

Crawler track assembly provides better stability, grater crowding force, lower ground bearing pressure (around 0.5 to 4 kg/cm^2). All the crawler tracks are presently independently propelled and their total number in a shovel varies from 2 to 8 depending upon the weight, size and type of the shovel. The steering is carried out by providing power to the crawler unit/units in one side and declutching the unit/units in the other side with the help of clutches and brakes. This action should be done by stopping the movement of the shovel, keeping the body in locking condition so that it cannot swing and also keeping the digging lock under locking condition. Larger capacity machines are mounted over four separate crawler units positioned in each corner and each having two endless track belts. This is done to control the mobility of the machine and to reduce the pressure over the ground. To control the undulation, the machine is mounted

over the crawler track with the help of four leveling jacks. The track belt made by the number of identical steel shoes. They are made endless by fastening with steel pins. Wedge shape projections are casted over the surface of the belt which are to be gripped by the bull wheel and idler and also the belt is supported and centered by the track rollers. Width and length of the crawler track varies depending upon the load and the working condition. Wider tracks are very good to lower ground bearing pressure, increasing machine stability but they are expensive, heavy, difficult during steering and is subjected to severe twisting stresses particularly in the undulated terrain. Longer crawler tracks increase machine stability and they are generally used for heavy load condition. As the bull wheel rotates the teeth will mesh with each shoe under and ahead of it and pushes the track belt and the idler forward. In higher capacity shovel crawler tracks are individually powered.

The crawler track assembly consists of steel crawler belts supported by a number of idlers held- on the truck frame and driven by a bull-wheel. The bull-wheels are made of alloy steel castings with lugs to engage the link castings. It is driven either by an independent motor unit or by a hydraulic prime mover through the reduction gearing box. In one of the versions the crawler steel belt is driven by the hoist motor through an electrically controlled air-actuated clutch mechanism, which is a duplicate of one used for hoisting purpose. The motor is connected to a combined splined shaft via. the said clutch. The crawler belt is made of flat shoes hinged and pinned together at their ends. In case of small capacity shovel, the pitch and width of the links are 370mm and 1.07m respectively. Diameter of the bull wheel is 1.05m. In one of the smaller capacity shovel model multi-jaw clutches are provided on the intermediate shafts of both the wheel axle for simultaneous or independent operation of the crawler track assemblies. The steel belt moves over the bull-wheel and passes around the idler sprocket and rollers. A screw-type jack is provided to tighten the slack belt by adjusting the position of the take-up unit which is guided in the crawler track frame unit. One end of a steel sleeve is connected to the centre of the truck frame and the other end is rigidly attached to the revolving frame. This sleeve is bored to accommodate bearings for the vertical propelling shaft. Operation. of the crawler track unit including the control of the steering clutches, locking of the crawler belt, etc. are governed from the operator's cab. Steering controlling is done by the manipulation of both the steering clutches and the brakes.

Crawler tracks support the main frame which contains a heavy swinging mechanism, counterweight platform, power unit (containing A. C. motor, hoist generator, swing generator, crowd generator, field generator), distribution box. transformer, motor and gearing system for hoisting, crowding, swinging, compressor for controlling system. fan operator's cabin with all controls and electric switches for proper controlling, steering, propelling, etc. The idler sprocket can be adjusted for tightening the track belt.

The revolving unit is built on a heavy steel bed supported by rollers which rest on a turn table. The turn table carries out the rotational motion. The turn table is mounted on a bull gear through a ball race which rotates by rotating a gear intermeshing with it. The gear is controlled by a set of bevel gears and clutches to enable the rotation of turn table either in clockwise or anticlockwise direction by respective clockwise and anticlockwise motor motion. The heavy shovels are generally provided with two swing gears which are independently driven by two motors one for each gear after suitably reducing the speed by reduction gearing system. Now-a-days magneto torque clutches are widely used in the power transmission system of the swinging mechanism.

The machine main body mounted over the revolving unit is welded construction (for smaller capacity shovel). or number of welded constructed units are bolted and/or riveted together to the main body.

The out rigger units (optional) are operated by the hydraulic pistons and generally provided at the rear end of the machine. It is used to stabilize the machine with the ground to withdraw the machine when crawler tracks are fastened with the muck and also to lift the back side of the machine during maintenance, repair of the track and other miscellaneous job leveling the machine, etc.

Magneto-torque

This may be considered as a 'magnetic coupling' which provide smooth power system and also control speed of a rotating system. It has high alloy steel fabricated inner and outer rotating members. The inner member is housed inside the outer member separated by an air gap. The outer member having sealed electrical winding rotates continuously by an A. C. motor or a diesel engine. The inner member consists of a circular steel rim. When the current is applied to the winding of the outer member, it is excited and produces magnetic field. Because of this reason a voltage is induced in the steel rim of the inner member and an eddy current flows in the rim. The inner member thereafter starts rotating in the same direction as the outer member. Higher the flow of D. C. current in the coil of the outer member higher would be the intensity of the magnetic field and hence it will enhance the rotational speed of the inner member. The r.p.m. of the inner

member also depends upon the load on it. This magentotorque supplies power without friction on outer members. The output characteristics of the system can be changed by changing the position of the controller which will energizes the coil of reverse unit and act as a smooth brake to stop the rotating body. This is widely used in the hoist mechanism in all electric shovel and also in swinging mechanism.

Operation and Efficiency of Shovel

The shovel should be kept in the safe distance from the face taking due care against the caving of the ledges from the high wall. The shovel should not be kept far way from the face. Otherwise it will require longer dipper reach, the front end rake of which get damages in the faster manner and also cycle time increases. This may also cause less loading in the bucket to prevent destabilization of the shovel body. Teeth of the bucket should be optimum in size and sharp enough to dig well into the bank. With the help of hoist and crowd mechanism the shovel is to be swung to the face. The bucket thereafter is lowered down until the teeth orient in horizontal position when the latch bar of the heavy bucket door will be held securely in the latch holder. The crowd mechanism moves the dipper stick downward through the saddle block till the bucket teeth touch the foot of the bank. The hoist mechanism moves the bucket forwards and downward and the crowd mechanism control the depth and angle of digging. Bucket should never be crowded very deep into the material which will delay the hoisting time. When the bucket will be hoisted up it will be totally filled up and after a little hoisting to the required height it is to be swung over the truck or railway wagon which are to be lined up with the arc of swing and as close to the shovel but at a safer distance. After slight crowding and releasing hoist cable a little bit ,the bucket is placed in proper position over the haulage unit. As soon as the trip cable is pulled, the

door of the bucket will open which will discharge the loaded material over the haulage unit. In case the high wall is very hard and digging is not very easy , the face rocks are to be dislodged with the help of empty bucket of the shovel. After loading is over the bucket is to be retracted slightly and there after is to be swung back to the digging position to repeat the cycle of operation. Shovels should always be moved over dry floor condition.

If wet and sticky floor is there they should be encountered by dry fills or putting wooden sleeper or steel plates over them. It is always desirable that a shovel should be operated smoothly and most effectively as per the guidelines given by the manufacturer of the machine. Direct excavation of soft rocks increases the efficiency of shovel because of higher diggability, more stable benches, formation or presence of less amount of lumpy boulders, selecting a definite boom angle for achieving a particular digging and dump height, digging and dumping radius, shifting of the shovel after cutting single or double slices (thickness of slice is around 0.3m to 0.8 m of more) in the faces, etc. In this case the normal longitudinal cuts of width is around 1.6 times the maximum digging radius of the shovel and is very good for truck rail transport system. But sometimes due to some factors the size of the cuts may be designed narrow or wide to increase the efficiency of the shovel.

The following precautions should be adopted during loading operations:

- A driver of any truck should not enter or leave his cab;
- The loaded bucket of any shovel or loader should not be traversed over the driver's cab of a truck or other motor vehicle;
- All persons should keep clear of the area between the loading unit and the mine face and the area traversed by the loading bucket during loading operations;
- All rocks that are too large to be handled safely should be broken before loading and transport equipment should be loaded in a manner to minimize spillage during haulage.

Presently a shovel have a bucket capacity of around 200 m^3. The largest size shovel may have a weight around 14000 tonne and total power more than 22000 kW whereas the production is around 17000 tonne per hour (if the density of the rock is taken around 2.5 t/m^3). Propelling speed of the shovel is around 1 to 3.5 km/hour. Efficiency of a shovel may be improved with the following practices-

- Using the maximum size of the bucket as per the capacity of shovel
- By selecting the appropriate height of the bench according to the height of the cut. Higher height bench will result caving of rock or ore from the top of the bench and hamper the shovel operation. So height of the bench should be equal to the maximum digging height of the shovel to control the slope angle and the tendency of caving in the face. If the height of the bench is very small as compared to the maximum dipper reach , the bucket will be filled up with many cutting run over the face. It will increase the bucket loading time and hence the capacity of the shovel will also reduce accordingly. Therefore the height of the bench should not be less then

the 70% to 75% of the height of a bench. The height of the bench which is blasted depends upon the geotechnical parameters of rock like the sizes of fragmented mass, cohesive forces of the blasted rock mass, free flowing character, swell factor, density, moisture content, stability, brittleness, looseness, compactness, broken ground condition and other physical properties. If the blasting fragments are smaller in sizes, free flowing in nature, has higher stability to rest in the bench, and has a very good slope stability , the height of the bench may be 1.2 to more than 1.6 times the maximum digging radius of a shovel and where poor fragmentation causes production of larger lumps which cave down frequently then the size of the bench should be equal or less to the maximum digging radius of a shovel. Higher the height of the bench, higher would be the production from the face. Sometimes a bench height of 1 to 3 m more as compared to the maximum digging height of the shovel is selected if the ore body or rock is quite competent. The angle of the boom (efficient angle is around 30° to more than 45°) plays a very important role to select the maximum digging and dumping reach of the dipper. An optimum dumping height and radius are essential to enhance the output of the power shovel digging efficiency, efficiency of the transporting system and to decrease the overall excavating, loading and transporting cost. Depending upon the height of the bench and the trucks or railway wagon, boom angle is fixed. Spotting of trucks or railway wagon depends upon this boom angle, adjustment of the crowd and hoist mechanism. So quick spotting of the transport equipment to a particular spot (keeping the shovel in it's particular point) to the location of the shovel is very important. A spotter may be employed to guide the truck operator to spot his truck at a particular spot. Accurate spotting at maximum time will definitely increase the shovel output. Sometime for overcasting operation the boom angle is changed in such a manner that the dipper gets its maximum digging reach and in that case there may be possibility of severe balancing problem of the shovel and hence in that case the capacity of the bucket is to be reduced to enhance shovel efficiency. Crowding of the bucket should be optimum to minimize the cycle of operation and to prevent the failure of rope.

- Angle of swing-The angle of swing is possible from 0° to 180°. More the swing angle to dump the material over truck or railway wagon or on spoil dump, more would be the necessity of cycle time to complete a cycle of operation, which will reduce the output capacity of the shovel. Swing angle generally should be around 45° .
- Shovel should be always within the safe distance, not near nor far off from the face. It is also dictated by the excavatability of the shovel and the type of haulage system.
- Cutting Depth-If the cutting depth is increased the bucket will be filled up only at one cutting run which will increase the output of the shovel. The depth of cut should be optimum to achieve maximum output by a shovel to a particular soil rock.
- Sticky material if present will stick inside the bucket, reduce the capacity and reduce the efficiency and output of a shovel. Likewise the big boulders if present due to bad fragmentation will hinder the loading and output of the shovel. Bucket fill factor varies from 40% to +100% depending upon size and shape of the bucket, the type, size, physical and chemical characteristics of ore or mineral, operator's skill, diggability of the bucket, overburden rock boulders etc. More the diggability into the mineral or rock, finer the size of the particle, less the moisture content and stickiness, higher would be the bucket fill factor. So bucket fill factor in case of easy digging materials like dry free flowing sand, loose and dry earth, smaller gravel is nearly 90% to 100% whereas it is around 40% to 60% in case of a very hard digging ,less fragmented, abrasive, big blocky rock boulders. Hence it is always advisable that the rock should be fragmented well for easy digging, crowding and for higher percentage of bucket fill factor.
- To minimizing the digging time it is always better if heap of ore or rock is piled up near to the shovel for quick and efficient digging.
- Use very good sharp teeth and bucket,
- Foggy weather, snow, rains etc. affect adversely the efficiency of a shovel and output.
- Good floor clearance by a front end loader or a bulldozer, good illumination, dry conditioned face, etc. will increase the efficiency and output of a shovel.
- Efficient supply and management of trucks, railway wagon for the consistency of operation of a shovel will increase the machine available time and hence will increase the output.
- Proper training, experience and skill of shovel operator, dumper driver, etc. will increase the efficiency and output of the shovel.

- Efficient repair and maintenance system will increase the operating efficiency and machine available time. A continuous operation will definitely yield more production.

Table 6.3 shows the specification of rope shovels

Table 6.3 Specifications of some shovels

Make	Model	Available bucket sizes (m^3)	Motor (kW)	Power Supply (v)	Dumping Height (m)	Boom length (m)	Ballast (t)	Working Weight (t)
Bucyrus Erie	195 BII DC	General purpose:13 Coal :23.7	Peak power :1104 Av. 15min demand 386-497	4160/ 7250 3ph , 60Hz	7.06	14.02	78.6	386.8
Bucyrus Erie	290 BII DC	General purpose: 16.82	Peak power :1430 Av. 15min demand 501-644	4160/ 7250 3ph , 60Hz	8.89	17.06	105	543
Bucyrus Erie	295BII I GTO	General purpose: 22.9	Peak power :1732 Av. 15min demand 430-603	4160/ 7250 3ph , 60Hz	8.92	18	170.5	739
Bucyrus Erie	395 BI GTO	General purpose: 29.8	Peak power :2152 Av. 15min demand 538-753	7200 3ph , 60Hz	9.6	19.51	192	939.5
Bucyrus Erie	495B GTO	General purpose: 40.5	Peak power :3280 Av. 15min demand 820-1148	7200 3ph , 60Hz	9.46	19.51	259.5	1067.1

HYDRAULIC SHOVEL

Hydraulic shovels (Fig - 6.3) are slowly replacing the conventional rope shovels. Because of their higher productivity, higher efficiency, higher machine availability, smoother and easier control; possibility of application in both the pushing and pulling forces, efficient, smooth and controlled digging and sumping, faster loading cycle, possessing fewer parts as compared to a conventional rope shovel, less maintenance cost, low down time, possibility of continuous health monitoring system, less bulkiness and weight compare to a conventional shovel, ability to selective mining at any desired horizon, ability to dress the high wall (face) etc. hydraulic shovels are gaining much popularity in modern surface mining operation. Some of the hydraulic excavators employs step less control of traction and it's speed which offer simple reversing and operating mechanism with high reduction speed ratio. The machine is also very much reliable in operation. The main operations in a hydraulic excavator are done by the hydraulic pressure.

Working Principle

The hydraulic pressure is generated by a prime mover i.e. by a diesel engine (most commonly used) or by an electric motor which drives number of, hydraulic motors via a gearbox, used to perform most of the working functions. The hydraulic pump generates fluid pressure to the valve bank which directs and controls the maintenance of fluid pressure in numerous hydraulic circuits for the operation of various hydraulic units with higher efficiency. The hydraulic power operates all the crowd mechanism, hoist mechanism, swinging mechanism; traction mechanism etc. Presently a hydraulic excavator having bucket capacity varying from 0.4m^3 to more than 34 m^3 are available in the market.

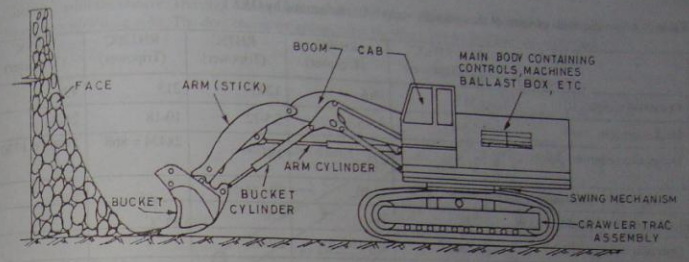


Fig. 6.3 : A Schematic Line Diagram of a Hydraulic Shovel Under Operation

Description

The arm and boom of a hydraulic shovel are of welded box design with high tensile steel having cast part of all pivot points. In Tripower version, the bearing points of boom/superstructure, boom/arm and arm/bucket are made of heavy cast steel parts to increase more stability. The lip of the bottom dump shovel is made of high-tensile fine grain steel in a robust welded box design. The rear wall of the bottom dump bucket is also made of high tensile structure through welded box design. The front dump bucket is made of high tensile fine-grain steel in all areas of potential wear and is of welded box design in the pivot areas. Both the bottom dump and front dump bucket have V-edges and high tensile drop-forged replaceable tooth tips supported by the socket - type mounting. In some version tooth are made of cast steel which are of self sharpening type. The hydraulic cylinders are made of seamless pipe materials having super fine and rolled inner surface. The piston rod is hardened, ground, surface treated and eventually chromium plated to enhance the life of guide bushings and head seals. Both side of hydraulic cylinders are attached with joint bearings. Swinging gear is operated by planetary gear mechanism with the help of hydraulic motor. The swinging is done at around 0 to 4.7 rpm.

Superstructure and undercarriage connection is provided through three path roller-bearing slewing ring with sealed internal toothing. Traveling mechanism is governed by crawler tracks, each of which is operated by individual hydraulic drives via an axial piston motor, each connected to a pinion planetary transmission. Travel speed and maximum tractive force are around 2 km/h to 2.1 km/h and 428 KN to more than 790 KN, respectively.

Besides all the controls in operator's cab, arrangements of automatic and constant angle bucket guidance during the horizontal crowd movement at any height and reach, during lifting and lowering of the attachment etc., automatic rollback of the bucket to prevent curling of the same, constant lift moment within the complete slewing range of the boom, etc. are provided. With the help of floating position a limited vertical reaction forces are transmitted to the basic machine during the crowding movement of the bucket. With this arrangement the control of the machine is made easier, the operating life of the machine is lengthened considerably and energy losses are prevented to a great extent. Wear at the bottom of the bucket also become minimum. Time relay controlled automatic lubrication to all pivot points on the attachment (except the pivot points between bucket and arm), the roller paths and toothing of the slewing ring are provided. Automatic centralized lubrication of all bearing points in a programmed adjustable time interval, controlled by a time relay is provided in the machine. A RH120C shovel has a cycle time around 25 s and can fill a 120t dumper with 5 buckets. A 170t dumper can be loaded in about 175s. With a bucket capacity of 12 m^3 and swell factor 1.3 it can excavate ore or rock at a rate of 1100 m^3/h. RH 40C shovel with the combination of 30t to 40 t dumpers (with a haul distance around 800 m) can produce limestone (hardness 1800 kg/cm^2- 2000 kg/cm^2) around 600t/h to 750 t/h.

Table 6.4 shows the specifications of some of the hydraulic shovels (manufactured by O&K)

Table 6.4 Specifications of some of the hydraulic shovels (manufactured by O&K)

	RH30C (Tripower)	RH40C (Tripower)	RH75C (Tripower)	RH120C (Tripower)	RH300C (Tripower)
Operating weight , (t)	58.5	80.6	120.5	213	498
Bucket capacity, (m^3)	3.5-4	3.6-5.3	5.5-12	10-18	21-34
Diesel engine power, (kW)	216	278	434	2x434 = 868	2x865 = 1730
Average fuel consumption, (l/hr)	30-35	45-50	70		
Electric motor (if employed),kW	200	240	420		
Operating voltage ,(KV)	0.5, 0.66	0.5, 0.66, 6.6	2.2-6.6		
Hydraulic output ,(kW)	2x84	239	399		
Amount of hydraulic pump flow, (l/min)	2x247	2x304	2x456		
Working pressure (bar)	upto 300	upto 300	upto 300		
Crawler track width (m)	0.6, 0.75, 0.9	0.7 and 1	0.75 and 1		
Crawler track length (m)	5.43	5.72	6.07		
Ground bearing pressure, (kg/cm^2)	1.06, 0.87, 0.74	1.18, 0.85	1.55, 1.18		

DRAG LINE

The total population of dragline in India is around 50 with a maximum capacity of bucket size is $30m^3$. A dragline is an excavator which has a boom (length varies form 9m to 96 m approx) one end of which is attached with the revolving unit of the machine and the hanging end in the other side carries a large sheave for the cable attached with the bucket. It is made of lattice construction by the structural steel which is lowered down or raised up by the cable of boom hoist. For dragging the bucket towards the machine one end of a drag cable is attached with the bucket and the other end is connected to the drag hoist via fairlead which is at the foot of the boom. The bucket is filled up by pulling or dragging the bucket against the loose material by the drag cable and then it is hoisted up by the hoist cable (Fig 6.5). Finally it dumps the material directly over the spoil dump or over the trucks or railway wagons. Generally the draglines are used for direct handling and rehandling of over burden material during overcastting since it is the cheapest means of overburden removal. It has longer reach and does not need auxiliary haulage equipment. It is also used to handle soft and unconsolidated materials, blasted rock or mineral, coal, top soil, rehandling of ore or coal stockpile, etc. Except direct casting for overburden removal it requires supporting haulage equipment. Since the dragline booms are longer in length is can dig well below and above the level where it stand and has larger flexibility in operating condition as compared to a shovel although it is 80% to 85% efficient in material handling as compared to an identical shovel. A dragline can nicely be used for stripping into the under-water, stripping of soft ore, sub-level operations, removing a high cut and stacking in one operation, to work in muddy and unstable condition, digging semi-consolidated soil, etc. But it can not work on a steep grade on the uneven footwall condition.

System of Working

The dragline may either be crawler mounted (most common), wagon mounted, track mounted or walking type. The boom can move both vertically (from horizontal), 25° to 60° (more common is 30° to 35°) and horizontally (0° to 180° with the help of swing mechanism) to perform the job. Rear end of the box shaped bucket whose one end is open, is attached by means of two hoist chains with the help of pins. The other end of the hoist chains are fitted with a dump sheave. To prevent the chains from rubbing on the side of the bucket a spreader bar is attached with the chains before

connecting with the sheave. The drag chains are attached in the front side of the bucket at one end while the other end is connected with a drag yoke. The drag chains are connected to the drag cable by dragline socket (Fig-6.6).

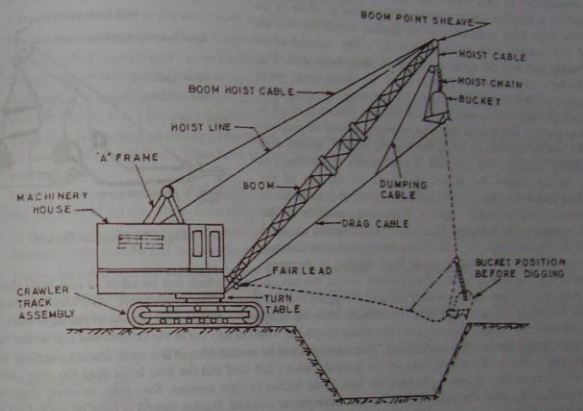


Fig. 6.5 : A Generalised Schematic Line Diagram of a Dragline Under Operation

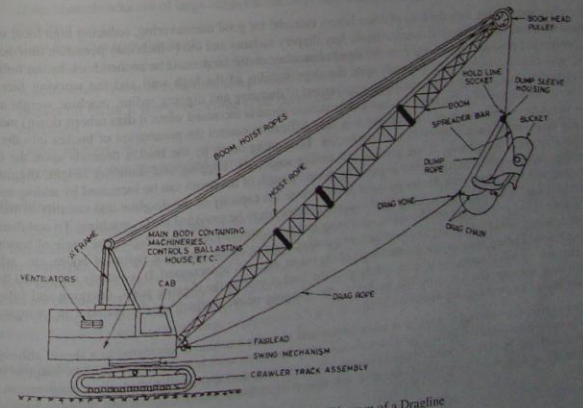


Fig. 6.6 : A Schematic Line Diagram of a Dragline

Direct digging is possible in case loose sandstone, soil, etc. are to be encountered where as blasting is necessary to the full extent prior to excavation. Bucket weight, design and balance together with the angularity of the drag cable (angle around 16° to 18°) forces, etc. are the main parameters of penetration of the bucket into the material being excavated by a dragline. Concentration of the forces are done by the replaceable type teeth attached with the bucket, which dig into the material at a steeper angle (around 18° to 22°). The material of the heavy bucket, lip and the teeth are made of manganese steel (Fig. 6.7). Buckets are sometimes perforated to drain out water accompanied with the collected muck. During dragging, drag cable should be kept out of touch with the muck pile and ground as much as possible to prevent much wear and also swinging should be properly controlled and minimized to avoid bend in the heavily loaded drag cable. During digging , the machine is periodically repositioned to optimize bucket control for improved filling, for shaping of the digging face and also to maximize reach for disposal of overburden or ore. Production will definitely improve if multiple passes to fill the bucket is avoided. if the bucket is filled by travel a distance around 2 to 3 times the bucket length, by minimizing the vertical hoist distance and also by minimizing the swing angle from digging to dumping.

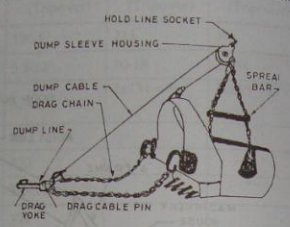


Fig. 6.7 : Dragline Bucket.

During swinging, the bucket gains a considerable inertia, the advantage of which can be taken to cast both the bucket material beyond the boom end increasing the actual digging and dumping radius respectively. At every advance of the dragline an initial key cut in the high wall is necessary while the dragline is located over the top of the high wall. In case of a dragline with 24m^3 bucket capacity and 90m boom length the width of key cut at the bottom and top of the bench of 40 m height is around 5m and 41m respectively . The cut width will be around 86m in this case. During digging, the hoist cable is slacked so that bucket totally rest over the ground with it's full load and the drag hoist drag the drag cable. The drag travel is reversed during hoisting to keep the load in the bucket in right manner. Swinging, hoisting (interact with each other) and acceleration after digging, require maximum power demand. Bucket should neither be pulled up into the fairlead or boom point, nor it should be lifted up into the boom (it is also called the 'Thight Lining'). Warning devices and mechanical and/or electrical motion cut-offs are to be incorporated in the machine to alert the machine operator and also to prevent this incidents.

Leveling and grading with the help of dozer is very essential for good maneuvering, reducing high local stresses on the ground floor and setting dragline over the bank. Any slippery surfaces and din (which may penetrate into the roller circle mechanism and produce heavy wear onto the rope) accumulate on the bank must be pushed back by the bulldozer. The resting spot of the dragline body depends upon the slope stability of the high wall and the working face, blast effected fragmented zone, level ground condition, requirement of dumping and digging radius, machine weight and it's physical size etc. Efficiency of a dragline is reduced and machine abuse is increased when it digs (chops down) materials from the working faces above its level. In that case it is very difficult to control the penetration of bucket into the muck and also high drag forces are necessary to pull the bucket. The cutting depth by the bucket depends upon the bucket weight, teeth sharpness, diggability of the ground, the cutting angle, etc. Digging and dumping height, digging and dumping radius depends upon the length and angle of the boom. Length of the boom can be increased by adding up extra section of structural steel work with the boom but it will decrease the capacity of the dragline and stability of the main machine body. To stabilize the dragline additional counter weights are to be provided to the machine. To stabilize large draglines double crawler tracks in both the sides, stabilizing jacks, etc. are also provided. Higher the angle of boom from the horizontal, lower would be the digging and dumping radius, digging depth, clearance radius of boom, etc., whereas the dumping height, casting distance, clearance height of the boom, etc. will be increased. It is already mentioned that the cycle of operation of a dragline consist of positioning of the bucket near to the digging point, dragging and filling the bucket, optimum hoisting and swinging the bucket to the dumping point, discharging the material in swinging and lowering the bucket for next digging operation.

For overburden removal a dragline is more mobile and versatile machine as compared to a shovel although it's loading efficiency is less. On the loosely ground surface some preparation work is necessary to keep the dragline over it for efficient working. In case of softer, deeper overburden in wet and geologically disturbed pit, a dragline can work more efficiently as compared to a shovel. Dragline can also efficiently be used for recovering minerals from placer deposits.

Selection of Dragline

The following points are to be considered during selection of a dragline-

- It is generally used for handling softer unconsolidated material,
- It has a greater ability to dig well above and below grade. It can excavate deep box cut.
- It has a greater digging reach and dumping radius,
- Since dragline boom length can be lengthened and shortened, its working range also can be varied according to desire, may or may not require other equipment for disposal,
- Although it is less efficient as compared to the shovel, it can function well under less rigid operating condition then the shovel, i. e. it has more flexibility in operation.
- Since they are located on the top of the bench, surface bench slides, water seepage, small undulation of surface (in case of large undulation, it has to be leveled first) does not hamper the effective operation of dragline during overcastting,
- During opening of opencast mine dragline is the most suitable excavator due to its larger reach and location at the top level of the cut. Multi-seam stripping is possible.
- In the soft mineral deposit the dragline can operate efficiently as compared to a shovel. The former can be located on the surface and load on the dumper on the surface itself.
- Heavy duty dragline are walking type and light duty one are of crawler mounted,
- They can negotiate a gradient up to 12° and ground pressure 1-2 kg/cm^2,
- In the presence of a hopper - reloader draglines can load material onto railway wagon, belt conveyor and other transport facilities,
- Height of waste heap is always restricted by the dragline dumping radius and not by the dumping height.
- Superior in wet pits,
- Maintenance is cheap,

The Dragline has following demerits also-

- If the blasted rocks are of large lump, the bucket is filled insufficiently and the bucket and drag rope wear rapidly,
- Used for softer rock formation
- Production cost is more as compared to the powered shovel,
- It's efficiency is less as compared to a powered shovel,
- Bucket fill factor is less as compared to the shovel and it requires well blasted/fragmented rocks to be excavated.
- Lesser spotting ability,
- Lesser output than the powered shovel,

The Machine

A dragline consists of bucket, boom ,power transmission unit for bucket operation, boom hoist mechanism, swing mechanism, propelling mechanism (crawler track mechanism, walking mechanism- mechanical eccentric type, hydraulic ram type), lubrication system, under carriage unit, etc. The booms are of lattice construction and made of low alloy steel tubes or steel angles. Longer length booms are supported by an 'A' -frame mast with the help of stay ropes. Draglines are powered by high capacity A. C. synchronous motor which in turn supply power to a hoist generator, a drag generator, a swing generator, etc. to supply D. C. power to the two hoist motor, two drag motor, two swing motor, etc. respectively. Supplied power voltage range is generally 3.3 KV to 6.6 KV.

Bucket Hoist Mechanism

The bucket hoist mechanism is meant for lowering or raising the bucket. It may be either 2 or 3 part hoist. In one of the design for smaller capacity dragline the motor provides power to the hoist drum via gearbox unit. The forged steel shaft of the gearbox unit connected to the hoist drum, mounts over the journal bearings. The hoist rope directed from the drum passes round a sheave at the A-frame apex, then over the sheave mounted at the top of the mast (when

fitted) over the sheave(s) at the boom point to the dump sheave case to form a 2 or 3 part hoist. The brake of the boom hoist drum is electrically operated and air released type. It is mounted over the motor armature shaft. This brake is used to control the bucket position and also immediately applied as soon as the power supply is off.

Fairlead

The purpose of the fairlead is to lead the drag rope nicely into the cable drum and also to not to allow rubbing or snapping of rope on the boom or other parts of the dragline. It is mounted at the front of the revolving frame at the foot of the boom. In one model the fairlead consists of 2 horizontal cast steel sheaves (which are adjustable for different sizes of drums) which allows leading of rope from the fairlead to the drum. The. fairlead has also 2 vertical sheaves mounted over the cast steel swiveling frame which swings in line with the drag rope irrespective of the position of the bucket.

Drag Mechanism

One end of the drag rope is connected to a drag drum unit and the other end is to the drag chain of the bucket by a dragline yoke. It's purpose is to handle (dragging) the bucket under the control of the drag controlled lever. In one of the small capacity dragline models, the drag motor unit is connected with the drag drum unit through an electrically controlled air actuated clutch. The drum is a split lagging type with machine - turned grooves for accommodating the drag rope. An electrically controlled brake is mounted over the primary pinion shaft.

Boom Unit

The boom unit is of lattice and cross-braced construction. All its chord members and lacings are made from structural steel and of welded construction. The lower portion of the boom unit is connected to the revolving unit. The sheaves are mounted at the boom point with the help of journal bearings. The sheaves are made from cast steel. In smaller capacity Dragline models ,each boom is made up of at least 2 sections but in case of higher capacity model additional intermediate sections (around 3m) are inserted in between the two end sections.

Boom Hoist Mechanism

The boom is lowered down or raised up by four part rope suspension system. It is similar to the shovel. The rope is coiled or uncoiled to a boom hoist rope drum which is operated by a separate motor via. a gearbox unit. In case of compact unit, this boom-hoist system is mounted on a structural support on the roof in the truss which is formed by the upper part of the ' A' -frame. In case of smaller dragline models, the rope is revved between the sheaves at the apex of the 'A' frame and at the boom point. But in case of larger dragline ,the boom is supported by an auxiliary mast-mounted at the foot of the boom. It also have a bridge-strand guys between the upper end and the boom point pin. The boom hoist rope is revved between the' A' -frame apex and' sheaves at the top of the mast.

Operator's Cab

The cab is substantially constructed by steel sheet and glass members and it is of self-supporting type. Removable panels and doors in them (for inspection and maintenance) are provided over the motor generator set, swinging and hoisting system, sides and rear end of the 'A'-frame. All the machinery are completely enclosed and the cab is provided with the doors and windows, auxiliary ventilating fan and also other ancilliary safety equipment. The cab of higher capacity dragline is completely air conditioned, heat, dust and noise proof. A panel incorporating all switches, consoles, etc. are located near to the operator's position for operating the machine. The driver seat is well designed .

WALKING DRAGLINE

The concept of a walking dragline is developed due to its following merits-

- It produces low ground pressure (around 0.3 to 0.9 kg/cm^2 depending upon size) because of it's larger base area.
- Turning of the machine is very easy and possible in a small space,
- It can do work in softer ground condition, in a slope around 15° and cable located very near to the bench edge.
- Requirement of power is very less,
- It does not damage the ground surface.
- It is very simple in construction. But their speed is very slow (around 200 to 500 m per hour depending upon size) and bulky in construction (since their width is more than 10 m, and hence it requires dismantling and reinstallation to shift it from one mine to another). It also cannot work efficiently in the foot of the high wall and requires prior leveling of ground during movement.

Table 6.5 shows a comparison among the crawler track unit , wheel unit and walking unit whereas the Table 6.6 shows the specifications of some of the walking Draglines.

Table 6.5 A general comparison among the crawler track unit , wheel unit and walking unit

Wheel unit	Crawler unit	Walking unit
1. Highly mobile	Moderately mobile	Less mobile
2. Damaging of road and the ground are less depending upon the type and size of the machine.	Damaging of road and the ground are more	Damaging of road and the ground are least
3. Stability of the machine is less	Stability of the machine is more	Stability of the machine is highest
4. Bearing pressure over the ground is high. More probability of tyre bogging down in the soft floor	Bearing pressure over the ground is moderate. Less probability of bogging down of crawler unit in the soft floor.	Bearing pressure over the ground is least. Least probability of bogging down of walking unit in the soft floor.
5. Traveling speed is high	Traveling speed is moderate	Traveling speed is least.
6. Offer less rolling resistance. Requirement of power is less as compared to an identical crawler unit.	Rolling resistance is high. Hence requirement of power is high	Power requirement to propel is very less.
7. It is efficient in the gradient less then 6°. However even sometimes 12° gradient negotiation is possible. Tractive effort plays the main significant role to negotiate gradient.	It is efficient in the gradient less than 12°. However sometimes even 15° to 16° gradient it can negotiate. Machine power plays the main significant role to negotiate gradient	It is efficient in the gradient less than 15°. However sometimes even 18° to 20° gradient it can negotiate. Hydraulic power or economic machine power play the main significant role to negotiate gradient.
8. It can travel longer distance economically.	It can travel shorter distance economically.	It's economic traveling distance is the shortest.
9. Less weight and bulkiness, and also less stable.	Medium weight and bulkiness and moderately stable.	Heavy weight and bulky and highly stable.
10. It can negotiate very rough and undulating surface.	It can negotiate moderately rough and undulating surface.	Require dismantling and reinstallation during area to area move. It requires plane surface to move.
11. Maneuverability and steering is less easy. It requires larger turning radius to turn.	Maneuverability and steering is easy. It requires less turning radius to turn.	Maneuverability and steering is very easy. It requires least turning radius to turn.
12. Simple in construction.	Complicated in construction.	Very simple in construction.
13. Tractive effort is less	Tractive effort is more	It is based on walking mechanism.
14. Can efficiently work near to the foot of the high wall.	Can efficiently work(near to the foot of the high wall.	Can not efficiently work near to foot of the high wall because of the machine's larger width.
15. Light duty to moderately heavy duty equipment are wheel mounted.	Moderately heavy to highly heavy duty equipment are crawler track mounted.	Heavy to super heavy duty equipment are walking unit mounted. Table 6.3 shows the specifications of some of the walking dragline.

TABLE 6.3
WALKING DRAGLINE SPECIFICATIONS

MAKE	MODEL	AVAILABLE BUCKET SIZES (cy yd)	MOTOR GENERATOR DRIVE MOTOR TOTAL HP	HOIST MOTOR HP	DRAG MOTOR HP	SWING MOTOR HP	PROPEL MOTOR HP	WEIGHT (lbs)	BALLAST (lbs)	APPROXIMATE RANGE OF MAX SUSPENDED LOADS (1000 lbs)	POWER SUPPLY (volts)
Bucyrus-Erie	280-W	9-16	1,000	–	–	250	–	916,000	116,450	45-80	4160
Bucyrus-Erie	480-W	12-18	1,000	1,045	–	390	–	1,545,000	260,000	62-82	4160
Bucyrus-Erie	680-W	14-26	1,600	1,500	1,080	660	660	1,804,000	200,000	70-130	–
Bucyrus-Erie	1260-W	24-40	2,450	2,090	2,090	1,500	750	3,335,000	450,000	116-200	4160
Bucyrus-Erie	1300-W	29-45	2,500	2,600	2,600	1,920	1,280	3,915,000	525,000	145-225	6900
Bucyrus-Erie	1350-W	37-48	4,000	3,200	3,200	2,560	1,280	5,743,000	650,000	185-225	6900
Bucyrus-Erie	1370-W	41-53	5,000	4,180	3,200	2,560,	1,280	6,076,900	900,000	230-265	6900
Bucyrus-Erie	1260-W	41-53	5,000	4,180	4,180	2,560	1,500	6,304,700	500,000	255-320	6900
Bucyrus-Erie	1370-W	51-62	6,000	5,200	5,200	3,200	2,000	7,087,700	850,000	285-400	6900
Bucyrus-Erie	1570-W	57-80	12,000	10,400	8,360	5,200	4,180	12,470,000	1,000,000	463-585	6900
Bucyrus-Erie	2570-W	93-115	18,000	15,600	10,400	7,800	7,800	17,435,000	1,800,000	915	–
Bucyrus-Erie	3270-W	125-176									
Marion	7450	9-17	n.a.	800	800	590	800	1,500,000	270,000	40-80	4160/7200
Marion	7500	13-22	1,500	1,300	1,300	510	1,300	1,580,000	350,000	67-108	–
Marion	7620	20-35	2,500	2,090	1,600	1,280	1,045	2,660,000	200,000	110-150	–
Marion	7820	25-45	3,000	2,600	2,600	1,500	1,300	3,860,000	400,000	185-210	–
Marion	8020	40-60	5,000	4,180	3,200	2,560	1,600	5,700,000	800,000	225-250	–
Marion	8050	50-60	5,000	4,180	4,180	2,560	1,600	6,500,000	900,000	280-320	–
Marion	8200	60-75	7,500	5,200	5,200	4,000	2,090	7,450,000	800,000	300-380	–
Marion	8750	75-120	13,000	10,400	7,800	6,270	3,200	13,050,000	1,600,000	400-583	–
Marion	8950	100-180	18,000	15,600	13,000	7,800	4,180	14,600,000	600,000	625-910	–
Page	732	11-20	1,250	1,045	1,045	510	500	1,933,000	340,000	55-100	4160/7200
Page	736	17-24	1,500	1,300	1,300	750	800	2,324,000	430,000	85-122	4160/7200
Page	740	18-30	2,550	2,090	2,090	1,125	1,045	3,184,000	450,000	90-150	4160/7200
Page	840	18-38	2,500	2,090	2,090	1,500	1,045	3,468,000	550,000	90-190	4160/7200
Page	752	24-45	3,000	2,600	2,600	1,500	1,300	4,306,000	580,000	120-225	4160/7200
Page	852	25-45	3,500	2,600	2,600	2,000	1,300	4,823,000	670,000	125-225	4160/7200
Page	852 LR	44-62	5,000	4,180	4,180	2,560	2,090	6,690,000	650,000	220-310	4160/7200
Page	757	40-70	6,000	5,200	5,200	3,200	2,090	7,523,000	860,000	246-350	7200
Ransomes & Rapier W-700		10-17	1,250	–	–	390	–	1,072,000	180,000	50-85	3300/6600
Ransomes & Rapier W-800		13-20	1,500	1,280	1,280	750	750	1,730,000	162,000	65-100	–
Ransomes & Rapier W-1300		22-35	–	2,090	2,090	1,500	1,000	2,840,000	170,000	115-175	–
Ransomes & Rapier W-2000		34-45	3,500	2,600	2,600	1,920	1,280	3,704,000	374,000	170-225	–
Ransomes & Rapier W-3000		56-70	6,000	5,200	5,200	3,200	2,560	6,682,000	828,000	280-350	–

The working principle of the crank or cam type walking mechanism is as following-

The machine working base has a circular tub which rest over the ground. The turntable is mounted over it. On either side of the machine there is either eccentric crank or cam mechanism along with walking shaft which are connected to walking frame and walking shoes.

The Working Mechanism

(i) In the 1st position both the tub of the machine and shoes rest over the ground.

(ii) The eccentric mechanism of the walking shaft which when start rotation in the clockwise, up to an angle 45^0 from the initial starting point the back side of the base will start rising and at the same time it will move backward. The shoes are rest over the ground and heavy weight will transfer on them. On further eccentric crank motion the machine body will move further backward and downward keeping the shoes rest over the ground and at an angle of 90^0 the total body will rest over the ground. After then there will be no further backward movement of the body.

(iii) When the eccentric crank shaft starts moving towards 270^0 the walking shoe will start rising and start moving backward while the body will remain under resting condition over the ground. On further rotation ,the walking shoe will move further backward and at 360^0 the shoe will rest over the ground. This way a complete cycle of operation of the walking mechanism is performed.

For turning the machine first the revolving body is turned to a desired angle and then the walking mechanism is started. The machine will thereafter turn accordingly.

Hydraulic Ram-Type (Fig - 6.8)

In this case there are two shoes (one in each side) and each shoes are attached with the upper deck of the machine body by two hydraulic cylinders. One cylinder is called the lifting cylinder, the other one the auxiliary cylinder. Both of them are connected to a transverse place attached with the shoe in such a manner that the shoe -can negotiate some undulation. Keeping the body in the resting condition over the ground, lift cylinders lift the shoe up by retracting the inner cylinder into the outer one and the tilt cylinder pushes the shoes towards the walking direction by extending the inner cylinder till the lifting cylinders orient in vertical position. Thereafter with the help of lifting cylinder the shoes are lowered down and pressurized (around 180 kg/cm^2) highly against the ground surface so that the total body of the dragline is lifted up. Afterward by retracting the inner cylinder of the auxiliary cylinder in side the outer cylinder the body is moved forward and lowered down again over the ground surface to complete the cycle of operation.

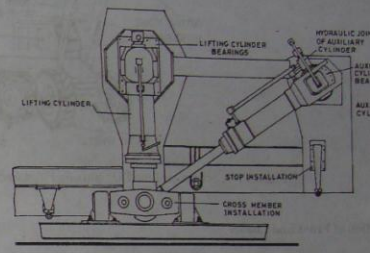


Fig. 6.8 : Walking Mechanism of a Dragline

Draglines are available in various capacities, bucket capacity ranging from 0.3 m^3 to 168 m^3 with boom length varying around ranging form 10 m to 96 m. Width and length of the walking shoe may vary from 1 m to 6 m and 6 m to 39 m (approx) respectively. A digging depth and dumping radius around 56 m and 92 m respectively are not quite uncommon. For softer rock and dumping over a haulage unit a dragline having capacity around 6 m^3 to 10 m^{3}and for overcastting a dragline around 10 m^3 to 30 m^3 are very much advantageous.

FRONT END LOADER

The front end loader is either a crawler track or wheel mounted tractor having a bucket at the front of the mainframe. It is used for loading coal or ore from a smaller height bench (say a height around 3m to 15m) onto the trucks, rail wagon or other conveying system. It also loads and transports material, ore or coal to a smaller distance. During stripping it piles up blasted rock mass or mineral to assist power shovel for loading, handling etc. It also spreads rock or ore in the stockpile of material handling station. The machine can nicely be used for the removal and hauling of top soil for a distance of 150m and a thickness of bench around 2m. The crawler track mounted front end loader is used where the ground condition is soft, terrain is rough and also where high digging force is required to excavate the loose or blasted rock mass or mineral. The wheel mounted front end loader has a very high degree of mobility. It is a highly speedy loading machine and has a high degree of flexibility and a high degree of productivity and longevity. The machine is highly efficient for unconsolidated rock, good fragmented rock mass or ore, but less efficient of handling blocky poor fragmented rock mass. It can be used efficiently on the soft to hard floor condition. It has a high digging capability and moderate to high production capacity. Combination of loading by front end loader and transporting by dumper can transport economically ore to a distance of more than 3 km for a very low to very high production rate.

For rehandling of spoil for a distance of 100m to 150 m it is one of the best machine. When a front-end-loader is not being used for loading coal/ore/ overburden rock, it can be used for the road construction, spot filling, floor cleaning around the shovel, lifting and transporting equipment/component of the equipment, etc.

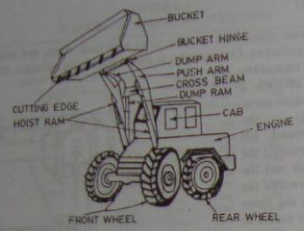


Fig. 6.9 : A Schematic Line Diagram of a Front End Loader (Wheel Mounted).

Selection of Front End Loader

It is to be done based on the following considerations:

- Where there are the existence of fluctuation of demand of ore / coal in the market
- For multi-bench or multi-pit operations
- Where blending of coal/ore is required
- Where capital to be invested for the procurement of equipment is less. A front-end-loader may work independently without any auxiliary or supplementary clean-up equipment like bulldozer.
- In case the life of the opencast mine is short.
- In case where higher production and less investment is desired.
- Where less skilled operating persons are required.
- Where there are the presence of moderately easy digging materials and well fragmented rocks.

The Bucket and Other Main Units

The cutting edge (made from manganese steel) of the bucket of the loader may be either straight type for maximum cleaning and loading efficiency and also for higher flexibility of operation whereas the 'vee' type cutting teeth are used for better penetration in the muck pile. Size of the bucket depends on the requirement of production capacity, cycle time, bucket fill factor, digging factor, operating efficiency, swell factor, machine availability, propel time factor, etc.

Based on the capacity of the bucket the size of the front-end- loader is selected. Sometimes counter weight is provided onto the loader for higher stability of the machine and to increase the operating load capability. The bucket may be either powered controlled or gravity controlled. The gravity controlled bucket has an arrangement to release the loaded bucket under gravity and also the bucket can be lifted up either by hydraulic mechanism or by rope pulley system. Power operated buckets are controlled by the hydraulic system. The efficiency of the loader is lowered down while digging the harder and coarser materials. Buckets are generally arranged overturning type.

Loader (wheel mounted)

Beside bucket the loader consists the units like engine, arms, power transmission system, undercarriage unit or frame (Fig 6.9).

Generally the power is provided to the loader driving wheel in case of the wheel mounted front end loader or to the driving sprocket in case of the crawler track mounted front end loader by the diesel engine via torque converter, reduction gear box, propeller shafts and differential gearing systems.

Mechanical linkage arms are actuated by the double acting hydraulic pistons.

Raising or lowering of the bucket, tilting, floating and holding of the same are governed by the hydraulic system which consists oil tank, oil pump, lift and tilt cylinders, relief and other control valve system etc.

The loader consists two types of brakes which are-

(a) Hydraulic operated main brakes and (b) Parking brakes.

Operation

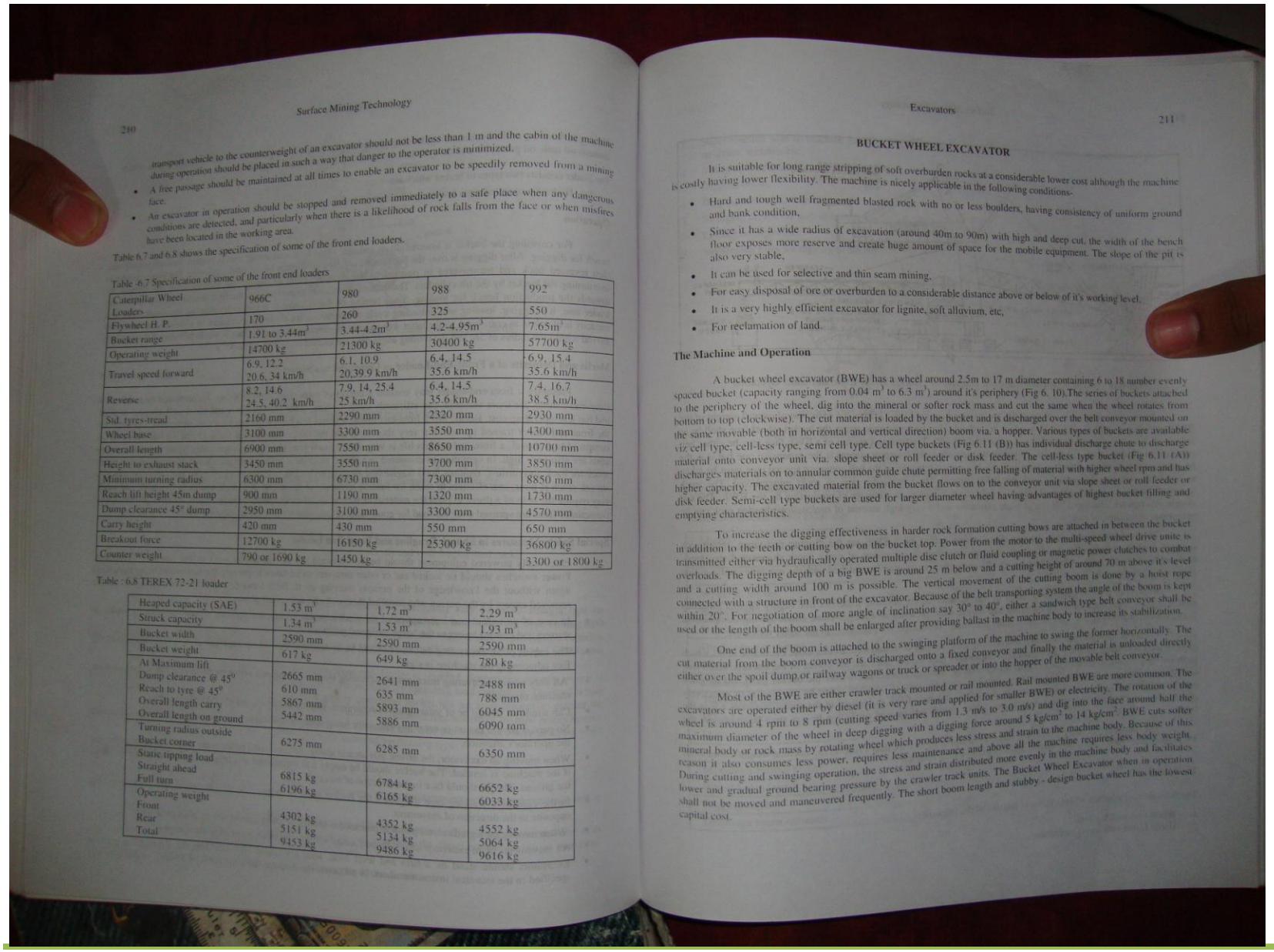
For crowding the bucket is lowered down and the whole front end loader unit is pushed forward towards the bench for digging. After digging is over the bucket unit is raised up slightly by the lifting cylinder. The loaded loader is then reversed back and maneuvered in position to the dumping point or unload the material over the dumper by overturning the bucket by the tilt cylinders. The loader is then again maneuvered to the right position and move forward towards the production bench for loading. Since turning of the machine is achieved by steering the whole front end loader unit, it's digging, loading and unloading cycle of operations are slower as compared to a shovel and hence the capacity is also less as compared to the latter. For gaining maximum loading efficiency it would be better if the loader is moved within the radius of 5m and the turning angle does not exceed more than 40° to 45°.

Merits and Demerits of a Front-End-Loader as Compared to a Shovel

Presently a front-end-loader may contain a bucket of more than $22m^3$ capacity. As compared to a shovel, a front-end-loaders are more versatile, can efficiently clean the face, grade the muck pile and blend well, cheaper in price and has less operating cost. Beside this a front-end-loader requires less capital intensive back-up equipment. Operators of the front-end-loader trained up within a very short time and the loader can be used for road making too. However as compared to a shovel, a front end loaders life is very short (5 to 6 years),has less salvage value, repair and maintenance costs are also very high. In operation as compared to a shovel machine it suffers form more shock loading, requires more space for back and turning, requires good fragmented rocks to be loaded, machine can not crowd and dig as smooth as a shovel, reach is poor, less efficient to reach for loading materials up to the end of a truck, bucket is very wide and weak, may require the help of a dozer to push the material as well as sometimes to push the loader itself, requires good roadway (sometimes fines are sprayed over the road for good padding) and also it's tyre life is also very small.

Special Safety Measures in Shovel ,Dragline and Front End Loader

- Electrically powered equipment should be disconnected before mechanical work is done on such equipment. Power switches should be locked out or other measures to be taken to prevent the equipment being connected again without the knowledge of the persons working on it. Such locks or protective devices should not be removed except by the person who installed them or by an authorized person (s) .
- In case of an unforeseen interruption of the electric power supply, the operator should immediately return all starters and control levers to the "stop" or "zero" positions.
- The cabins of all excavating machines should be so equipped that the operator can always keep that part of the face adjacent to the machine in sight.
- All sites where excavating machines are operating should be equipped with a means of communication with the machine operator.
- Cab windows should be of safety glass or equivalent, in good condition and should be kept clean.
- No person should mount or enter any excavator being operated and capable of movement without first attracting the operator's attention.
- When moving an excavator, the driven axle should always be positioned either at the downhill end or at the rear if the machine is leveled. The bucket should be empty and maintained at a height of not more than 1 m above the ground. The jib should face in the direction of motion.
- In the case of a walking type dragline , the bucket must be empty and the jib should face backwards, i.e. opposite to the direction of movement.
- When moving on a gradient, all possible precautions should be taken against sliding of an excavator.
- All movements of an excavator should be performed with the aid of an assistant to the operator.
- Excavators should stand on a firm and level base, with a gradient not exceeding the permissible gradient specified in the technical instruction sheet. In all cases, the distance from the flank of a bench or dump or any



transport vehicle to the counterweight of an excavator should not be less than 1 m and the cabin of the machine during operation should be placed in such a way that danger to the operator is minimized.

- A free passage should be maintained at all times to enable an excavator to be speedily removed from a mining face.
- An excavator in operation should be stopped and removed immediately to a safe place when any dangerous conditions are detected, and particularly when there is a likelihood of rock falls from the face or when misfires have been located in the working area.

Table 6.7 and 6.8 shows the specification of some of the front end loaders.

Table :6.7 Specification of some of the front end loaders

Caterpillar Wheel Loaders	966C	980	988	992
Flywheel H. P.	170	260	325	550
Bucket range	1.91 to 3.44m^3	3.44-4.2m^3	4.2-4.95m^3	7.65m^3
Operating weight	14700 kg	21300 kg	30400 kg	57700 kg
Travel speed forward	6.9, 12.2 20.6, 34 km/h	6.1, 10.9 20,39.9 km/h	6.4, 14.5 35.6 km/h	6.9, 15.4 35.6 km/h
Reverse	8.2, 14.6 24.5, 40.2 km/h	7.9, 14, 25.4 25 km/h	6.4, 14.5 35.6 km/h	7.4, 16.7 38.5 km/h
Std. tyres-tread	2160 mm	2290 mm	2320 mm	2930 mm
Wheel base	3100 mm	3300 mm	3550 mm	4300 mm
Overall length	6900 mm	7550 mm	8650 mm	10700 mm
Height to exhaust stack	3450 mm	3550 mm	3700 mm	3850 mm
Minimum turning radius	6300 mm	6730 mm	7300 mm	8850 mm
Reach lift height 45in dump	900 mm	1190 mm	1320 mm	1730 mm
Dump clearance 45° dump	2950 mm	3100 mm	3300 mm	4570 mm
Carry height	420 mm	430 mm	550 mm	650 mm
Breakout force	12700 kg	16150 kg	25300 kg	36800 kg
Counter weight	790 or 1690 kg	1450 kg	-	3300 or 1800 kg

Table : 6.8 TEREX 72-21 loader

Heaped capacity (SAE)	1.53 m^3	1.72 m^3	2.29 m^3
Struck capacity	1.34 m^3	1.53 m^3	1.93 m^3
Bucket width	2590 mm	2590 mm	2590 mm
Bucket weight	617 kg	649 kg	780 kg
At Maximum lift Dump clearance @ 45° Reach to tyre @ 45° Overall length carry Overall length on ground	2665 mm 610 mm 5867 mm 5442 mm	2641 mm 635 mm 5893 mm 5886 mm	2488 mm 788 mm 6045 mm 6090 mm
Turning radius outside Bucket corner	6275 mm	6285 mm	6350 mm
Static tipping load Straight ahead Full turn	6815 kg 6196 kg	6784 kg 6165 kg	6652 kg 6033 kg
Operating weight Front Rear Total	4302 kg 5151 kg 9453 kg	4352 kg 5134 kg 9486 kg	4552 kg 5064 kg 9616 kg

BUCKET WHEEL EXCAVATOR

It is suitable for long range stripping of soft overburden rocks at a considerable lower cost although the machine is costly having lower flexibility. The machine is nicely applicable in the following conditions-

- Hard and tough well fragmented blasted rock with no or less boulders, having consistency of uniform ground and bank condition,
- Since it has a wide radius of excavation (around 40m to 90m) with high and deep cut, the width of the bench floor exposes more reserve and create huge amount of space for the mobile equipment. The slope of the pit is also very stable,
- It can be used for selective and thin seam mining.
- For easy disposal of ore or overburden to a considerable distance above or below of it's working level.
- It is a very highly efficient excavator for lignite, soft alluvium, etc,
- For reclamation of land.

The Machine and Operation

A bucket wheel excavator (BWE) has a wheel around 2.5m to 17 m diameter containing 6 to 18 number evenly spaced bucket (capacity ranging from 0.04 m^3 to 6.3 m^3) around it's periphery (Fig 6. 10).The series of buckets attached to the periphery of the wheel, dig into the mineral or softer rock mass and cut the same when the wheel rotates from bottom to top (clockwise). The cut material is loaded by the bucket and is discharged over the belt conveyor mounted on the same movable (both in horizontal and vertical direction) boom via, a hopper. Various types of buckets are available viz cell type, cell-less type, semi cell type. Cell type buckets (Fig 6.11 (B)) has individual discharge chute to discharge material onto conveyor unit via, slope sheet or roll feeder or disk feeder. The cell-less type bucket (Fig 6.11 (A)) discharges materials on to annular common guide chute permitting free falling of material with higher wheel rpm and has higher capacity. The excavated material from the bucket flows on to the conveyor unit via slope sheet or roll feeder or disk feeder. Semi-cell type buckets are used for larger diameter wheel having advantages of highest bucket filling and emptying characteristics.

To increase the digging effectiveness in harder rock formation cutting bows are attached in between the bucket in addition to the teeth or cutting bow on the bucket top. Power from the motor to the multi-speed wheel drive units is transmitted either via hydraulically operated multiple disc clutch or fluid coupling or magnetic power clutches to combat overloads. The digging depth of a big BWE is around 25 m below and a cutting height of around 70 m above it's level and a cutting width around 100 m is possible. The vertical movement of the cutting boom is done by a hoist rope connected with a structure in front of the excavator. Because of the belt transporting system the angle of the boom is kept within 20°. For negotiation of more angle of inclination say 30° to 40°, either a sandwich type belt conveyor shall be used or the length of the boom shall be enlarged after providing ballast in the machine body to increase its stabilization.

One end of the boom is attached to the swinging platform of the machine to swing the former horizontally. The cut material from the boom conveyor is discharged onto a fixed conveyor and finally the material is unloaded directly either over the spoil dump or railway wagons or truck or spreader or into the hopper of the movable belt conveyor.

Most of the BWE are either crawler track mounted or rail mounted. Rail mounted BWE are more common. The excavators are operated either by diesel (it is very rare and applied for smaller BWE) or electricity. The rotation of the wheel is around 4 rpm to 8 rpm (cutting speed varies from 1.3 m/s to 3.0 m/s) and dig into the face around half the maximum diameter of the wheel in deep digging with a digging force around 5 kg/cm^2 to 14 kg/cm^2. BWE cuts softer mineral body or rock mass by rotating wheel which produces less stress and strain to the machine body. Because of this reason it also consumes less power, requires less maintenance and above all the machine requires less body weight. During cutting and swinging operation, the stress and strain distributed more evenly in the machine body and facilitates lower and gradual ground bearing pressure by the crawler track units. The Bucket Wheel Excavator when in operation shall not be moved and maneuvered frequently. The short boom length and stubby - design bucket wheel has the lowest capital cost.

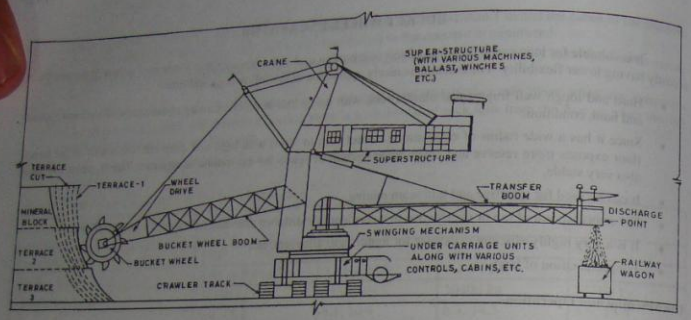


Fig. 6.10 : A Generalised Schematic line Diagram of a Bucket Wheel Excavator under Operation

An one unit reduction in weight of the bucket wheel causes 11 to 12 times the reduction of overall weight of the BWE and hence the cost of machine also reduce accordingly. For best economic return the bucket-wheel face shall be as long as possible with a permissible mining slope around 6° to 9°. The transport system of a BWE face may be either belt conveyor, truck dispatch or rail system. Although the belt conveyor transport system is the expensive one but it is the best option for mine transport for long term projects. But during selection of the belt conveyor special care must be taken to match the belt conveyor capacity with the BWE capacity for a particular type of rock/ mineral/coal. The truck transport system is a discrete system. Delay of supply of truck will cause stoppage of production by the BWE. The rail system although not continuous but if it is made automatic and a very high amount of ore/coal/overburden rock can be transported at the cheapest rate.

The basic parameters are to be considered during cutting by a BWE are the digging resistance, intact rock strength, jointing of bedding planes, fractures ,other geo-mechanical parameters of rocks, tooth shapes and sharpness, angle of attack, wedge angle of the cutting lip, etc. All these parameters must be matched with the machine configuration and power of the machine.

Buckets of a BWE adversely affected by the presence of the abrasive materials like highly angular siliceous sand and gravels, presence of sticky materials like plastic clay, presence of stronger stone bands having compressive strength more than 600 kg/cm^2, presence of smaller percentage of stronger rock boulders in the softer rocks, etc.

In case of stronger rock condition drop cutting method shall be introduced. In this case, the major items which shall be incorporated in the BWE are higher concentration of teeth on the sides and the corner of the buckets, scarifying teeth on the pre-cutters, parallel rather than off-set wheels, manganese steel liners over the bucket teeth and tungsten carbide over the tip of the teeth. The stronger stone bands thicker than 10 cm shall either be pre-blasted or selectively ripped and afterward they shall be excavated by the BWE. In case a BWE to be used in the sticky materials e.g in the plastic clay, either chain backed bucket or linatex rubber linings and sprayed on Teflon linings shall be used over the bucket.

The rate of production by the BWE varies from 100 m^3/h to more than 11000 m^3/h, machine weight varies from 35t to more than 7000 t and power around 200 kW to more than 7000 kW.

The Main Units

The main units of a bucket wheel excavator consists of the following-

1. Wheel unit with buckets,
2. Under frame with travel mechanism,

3. Superstructure with all machine housing along with motors, gearboxes and other mechanism, counter weight structure, winches etc.,
4. Swinging mechanism,
5. Digging boom,
6. Digging boom belt conveyor,
7. Swinging cantilever with discharge boom belt conveyor and chute,
8. Luffing arrangement,
9. Suspension truss for digging boom etc.

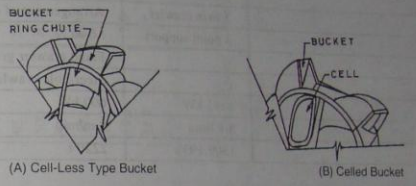


Fig. 6.11 : Schematic Line Diagrams of Type of Buckets of Bucket Wheel Excavator

Table 6.9, 6.10 and 6.11 show the specification of various types of Bucket Wheel Excavators whereas the table 6.12 shows the comparison among shovel , dragline, front end loader and bucket wheel excavator

Table : 6.9 Bucket wheel excavator

Mine	Neyveli Lignite Corporation Ltd.	
Manufacturer	B/W Bucket Wheel Excavator	
	I	II
Wheel diameter	8.4m	10.5m
Bucket capacity	700 liters	1400 liters
Number of buckets	9	10
Discharge per minute	65	65
Wheel r.p.m .	7.19	6.5
Cutting speed	3.16 m/s	3.57 m/s
Theoretical output	2730 m^3 /h	5460 m^3 /h
Wheel drive		
Main drive	750 kW	2x750 kW
Auxiliary drive	20 kW	20 kW
Maximum cutting height	20 m	27 m
Wheel boom length	26.3 m	31 m
Digging depth	3.0 m	-
Discharge boom outreach	35 m	35 m

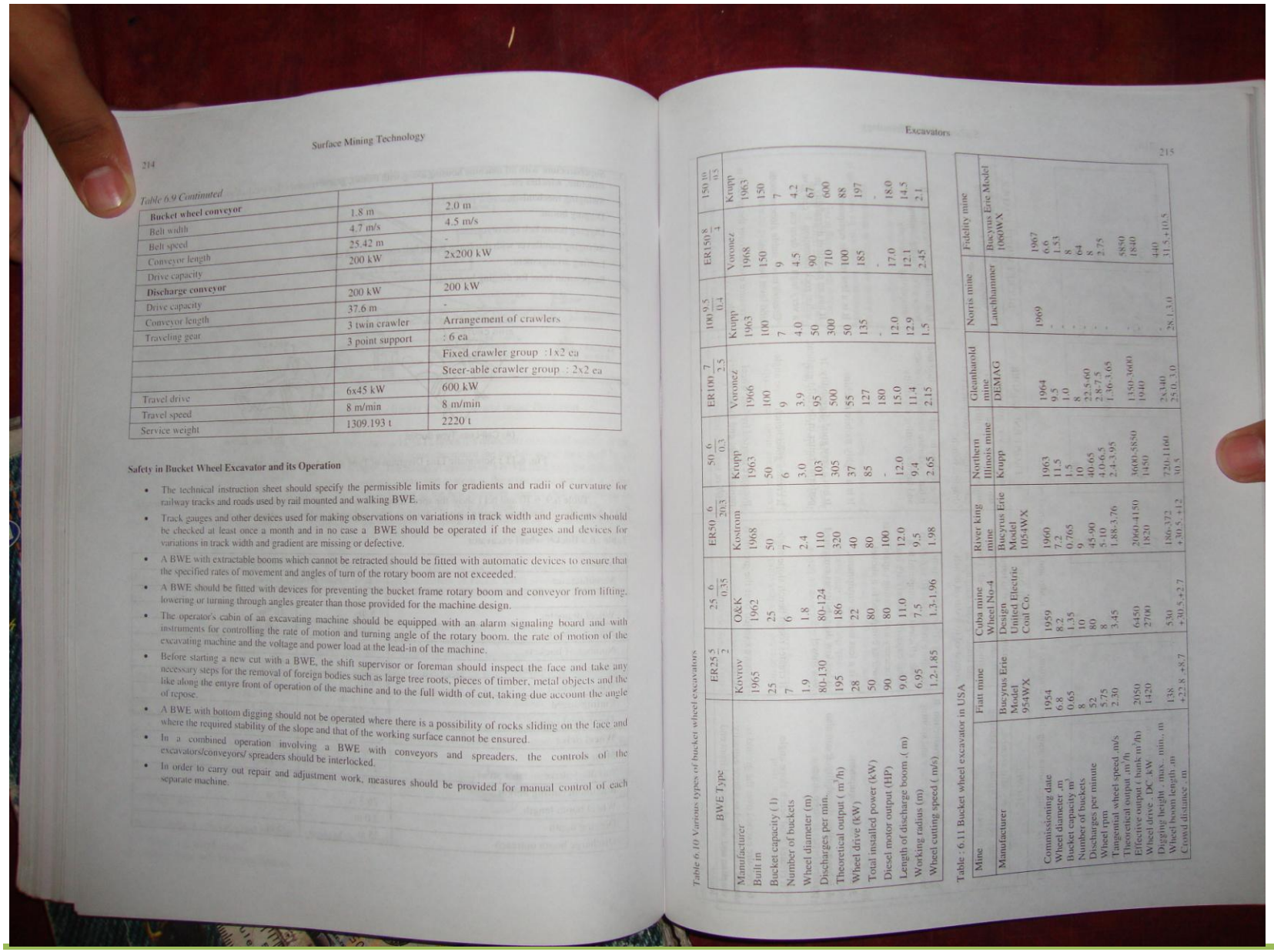


Table 6.9 Continued

Bucket wheel conveyor		
Belt width	1.8 m	2.0 m
Belt speed	4.7 m/s	4.5 m/s
Conveyor length	25.42 m	-
Drive capacity	200 kW	2x200 kW
Discharge conveyor		
Drive capacity	200 kW	200 kW
Conveyor length	37.6 m	-
Traveling gear	3 twin crawler	Arrangement of crawlers
	3 point support	: 6 ea
		Fixed crawler group : 1x2 ea
		Steer-able crawler group : 2x2 ea
Travel drive	6x45 kW	600 kW
Travel speed	8 m/min	8 m/min
Service weight	1309.193 t	2220 t

Safety in Bucket Wheel Excavator and its Operation

- The technical instruction sheet should specify the permissible limits for gradients and radii of curvature for railway tracks and roads used by rail mounted and walking BWE.
- Track gauges and other devices used for making observations on variations in track width and gradients should be checked at least once a month and in no case a BWE should be operated if the gauges and devices for variations in track width and gradient are missing or defective.
- A BWE with extractable booms which cannot be retracted should be fitted with automatic devices to ensure that the specified rates of movement and angles of turn of the rotary boom are not exceeded.
- A BWE should be fitted with devices for preventing the bucket frame rotary boom and conveyor from lifting, lowering or turning through angles greater than those provided for the machine design.
- The operator's cabin of an excavating machine should be equipped with an alarm signaling board and with instruments for controlling the rate of motion and turning angle of the rotary boom, the rate of motion of the excavating machine and the voltage and power load at the lead-in of the machine.
- Before starting a new cut with a BWE, the shift supervisor or foreman should inspect the face and take any necessary steps for the removal of foreign bodies such as large tree roots, pieces of timber, metal objects and the like along the entyre front of operation of the machine and to the full width of cut, taking due account the angle of repose.
- A BWE with bottom digging should not be operated where there is a possibility of rocks sliding on the face and where the required stability of the slope and that of the working surface cannot be ensured.
- In a combined operation involving a BWE with conveyors and spreaders, the controls of the excavators/conveyors/ spreaders should be interlocked.
- In order to carry out repair and adjustment work, measures should be provided for manual control of each separate machine.

Table 6.10 Various types of bucket wheel excavators

BWE Type	ER25 $\frac{5}{2}$	25 $\frac{6}{0.35}$	ER50 $\frac{6}{20.3}$	50 $\frac{6}{0.3}$	ER100 $\frac{7}{2.5}$	100 $\frac{9.5}{0.4}$	ER150 $\frac{8}{4}$	150 $\frac{10}{0.5}$
Manufacturer	Kovrov	O&K	Kostrom	Krupp	Voronez	Krupp	Voronez	Krupp
Built in	1965	1962	1968	1963	1966	1963	1968	1963
Bucket capacity (l)	25	25	50	50	100	100	150	150
Number of buckets	7	6	7	6	9	7	9	7
Wheel diameter (m)	1.9	1.8	2.4	3.0	3.9	4.0	4.5	4.2
Discharges per min.	80-130	80-124	110	103	95	50	90	67
Theoretical output (m^3/h)	195	186	320	305	500	300	710	600
Wheel drive (kW)	28	22	40	37	55	50	100	88
Total installed power (kW)	50	80	80	85	127	135	185	197
Diesel motor output (HP)	90	80	100	-	180	-	-	-
Length of discharge boom (m)	9.0	11.0	12.0	12.0	15.0	12.0	17.0	18.0
Working radius (m)	6.95	7,5	9.5	9.4	11.4	12.9	12.1	14.5
Wheel cutting speed (m/s)	1.2-1.85	1.3-1.96	1.98	2.65	2.15	1.5	2.45	2.1

Table : 6.11 Bucket wheel excavator in USA

Mine	Fratt mine	Cuba mine Wheel No-4	River king mine	Northern Illinois mine	Gleanburold mine	Norris mine	Fidelity mine
Manufacturer	Bucyrus Erie	Design	Bucyrus Erie	Krupp	DEMAG	Lauchhammer	Bucyrus Erie Model
	Model 954WX	United Electric Coal Co.	Model 1054WX				1060WX
Commissioning date	1954	1959	1960	1963	1964	1969	1967
Wheel diameter, m	6.8	8.2	7.2	11.5	9.5	-	6.6
Bucket capacity m^3	0.65	1.35	0.765	1.5	1.0	-	1.53
Number of buckets	8	10	9	10	8	-	8
Discharges per minute	52	80	45-90	40-65	22.5-60	-	64
Wheel rpm	5.75	8	5.10	4.0-6.5	2.8-7.5	-	2.75
Tangential wheel speed ,m/s	2.30	3.45	1.88-3.76	2.4-3.95	1.36-3.65	-	-
Theoretical output, m^3/h	2050	6450	2060-4150	3600-5850	1350-3680	-	5850
Effective output (bunk m^3/h)	1420	2700	1820	1450	1940	-	1840
Wheel drive , DC kW						-	440
Digging height , max. min., m	138	530	186-372	720-1160	2430	28 1.3.0	315.+10.5
Wheel boom length ,m	+27.8 +8.7	+30.5 +2.7	+30.5 +12	30.5	250.3.0		
Crowd distance ,m							

Table : 6.11 Continued

Transport distance, m	38.0	47.5	41.5	37.0	42.0	48.3	49.0
Wheel boom conveyor width, m	10.5	9.6	13.3	-	-	-	18.0
Discharge conveyor speed ,m/s	106	128	119	142.5	120.0	142.3	128.0
Service weight , short ton	1.35	1.52	1.35	2.20	1.40	-	1.50
Roller race diameter, m	5.0	4.6-6.3	5.0	5.2	6.0	-	5.5
	1250	2100	1530	3400	2200	2700	1470
	10.33	13.50	10.4	11.58	-	-	-

Table : 6.12 A comparison among shovel , dragline, front end loader and bucket wheel excavator

SHOVEL	DRAGLINE	FRONT END LOADER	BUCKET WHEEL LOADER
1. It can handle very hard abrasive, big lumps of rock. Drilling and blasting cost is the lowest.	Generally it is used to handle wet, soft, unconsolidated and also well fragmented blasted rock. Drilling and blasting cost is higher as compared to a shovel bench.	It can handle very soft to very hard ,abrasive, medium sized lumps of rock. Drilling and blasting cost is higher as compared to a dragline bench.	It is generally used to handle soft, unconsolidated and easy digging rocks or minerals.
2. Used for very high rate of production. It has higher loading efficiency and spotting capability	It is less efficient equipment compare to a shovel (It is 80% to 90%, efficient as compared to a same capacity shovel). It's loading efficiency and spotting capability is lower as compared to a shovel.	Use for low to high production rate. It has moderate loading efficiency and higher spotting capability.	Use for high production rate. It has a good cutting loading and transporting efficiency.
3. It is a less mobile equipment.	It is a less mobile equipment.	It is a highly mobile machine.	It is a less mobile equipment.
4. It has excellent diggability. It can also perform better in steep slope.	It has low diggability as compared to a shovel	It has a very good diggability. It performs better in the mild gradient.	It has an excellent diggability in softer rock formation. It can give a very high and deep cut.
5. It creates good bench slope angle.	It creates rough bench slope angle.	It creates good bench slope angle.	Creates more stable slope smooth and level benches.
6. Pressure exerted over the ground is lower as compared to a same capacity dragline and wheel mounted front end loader.	Pressure exerted over the ground is more.	Pressure exerted by four wheel mounted front end loader is very high.	Pressure exerted over the ground is high.

Table : 6.12 Continued

7. In most of the cases it requires auxiliary material handling system.	It may or may not require auxiliary supporting material handling system	It may or may not require auxiliary supporting material handling system.	It requires auxiliary material handling system.
8. It is limited to somewhat rigid operating condition.	It can operate under less rigid operating condition.	It can operate under very flexible operating condition.	It operates under rigid structural condition.
9. Digging above or below grade is limited. Over casting to a limited range is possible.	It can nicely dig above or below grade. Over-casting is generally done with the help of this machine.	It can dig and also transport load. Over-casting is not possible.	It can nicely dig above or below grade and deliver material above or below the working level.
10. Higher maintenance cost.	Maintenance cost is less as compared to a shovel.	Low maintenance cost.	Maintenance cost will be very high if hard rock intrusion is negotiated.
11. It is used for discrete operation.	It is used for discrete operation.	It is used for discrete operation.	It is used for continuous operation.
12. Selective mining is only possible if the bench height is properly designed.	Selective mining is generally not done with it.	Selective mining is only possible if bench height is properly designed.	Selective mining of inter bedded lignite or soft ore is possible.
13. It is a moderately expensive machine and cost is less as compared to a same capacity dragline or bucket wheel excavator.	It is a very expensive machine.	It is the least expensive machine among four.	It is very expensive machine.
14. Higher instantaneous demand of power.	Higher instantaneous demand of power.	Moderate instantaneous demand of power.	Lower instantaneous demand of power.
15. Loading can be done with shock over dumper.	Loading can be done with shock over dumper.	Loading is done with shock over dumper.	Loading is not done with shock on any transport system.
16. For higher output machine weight is heavy.	For higher output machine weight is heavy.	Low to high output is possible. weight is moderate.	For higher output machine weight is less.
17. For the same output the physical size of the equipment is less bulky as compared to a dragline.	For the same output the physical size of the equipment is bulky as compared to all the other three equipment.	For the same output the physical size of the equipment is smallest as compared to the other three equipment.	For the same output the physical size of the equipment is bulky as compared to the shovel and the front end loader.
18. Lower dumping radius and digging reach as compared to a same capacity dragline	Very high dumping radius and digging reach.	Lower dumping radius and digging reach compare to a same capacity shovel	Higher dumping radius with auxiliary material handling system.

Table : 6.13 Continued

19. Since the shovel is located over the top of the mineral bench, water, rough terrain, spalling of rock or ore or mineral from the bench slope, etc. affect the operation of a shovel. If the mineral is softer then mineral bench may not withstand the pressure exerted by the shovel on it.	Since the dragline is located over the top of the bench or on the surface which is properly leveled, water, rough terrain, spalling of rock or ore or mineral from the bench slope, etc. do not hamper the operation of a dragline. If the mineral is softer compare to the rock of the overburden, it can nicely be used.	Since the front end loader is also located over the top of the mineral bench, water, rough terrain, spalling of rock or ore or mineral from the bench slope, etc. affect the operation of a front end loader. If the mineral is softer than a very light weight front end loader should be used.	It has a higher cutting reach. Since the BWE is located to a particular position and it has a long boom, water, rough terrain, spalling of rock or ore or mineral from the bench slope or edge, etc. do not affect the operation of it. In a very soft and weak rock condition also it can nicely be used.
20. It has lower working reach as compared to the dragline. Hence for making box cut or while on development purpose, it is inferior as compared to a dragline. Working range is more or less rigid.	Since it has a higher reach it can be used for making box cut or development work more easily as compared to a shovel or a front end loader. Working range can be varied by varying the length of the boom.	It has lower working reach as compared to a dragline and a shovel. Hence during making box cut or while on development it is inferior compare to either a dragline or a shovel. Working range is more or less rigid.	Since it has a higher working reach it can nicely be used for development purposes, working range can be varied by varying the length of the boom.
21. Generally shovels are not used for reclamation work.	The draglines are efficiently used for reclamation work too.	Generally front end loaders are not used for reclamation work.	Generally the bucket wheel excavator are not used for reclamation work. However a Bucket Wheel Reclaimer is more or less similar to a BWE.

Dragline

Bucket wheel excavator

BACKHOE

It is also named as 'Pull Shovel', 'Drag Shovel' etc. It digs soil, rock or ore below the level of a bench (or bank) on which it stands and unload excavated materials over a truck or railway wagon. It can be deployed efficiently where the mine is very much wetted by the prolonged rain or where seepage of water through the ground strata is very heavy. Generally the backhoe is very good for trenching, shallow depth cutting and for basement excavations. The hoe bucket is attached to a dipper stick at it's lower end facing towards the equipment (Fig. 6.12). The middle of the dipper stick is hinged to a boom, where as the top of the same is attached with a rope pulley. The pulley is connected by a hoist rope to a jack boom attached to the main equipment body. A drag rope is connected to a drag drum passing through the sheave attached to the side of the main boom. The dipper is lowered down into position by the hoist rope so that bucket bites into ground and the cutting operation is achieved by pulling the drag rope till it becomes full. Afterward both the hoist and drag ropes are wind up to make the bucket more closer to the toe of the bank and thereafter bucket is raised up to the dumping height by operating the hoist rope. The equipment is swung to unload the material over a dumper or a railway wagon or other haulage system by paying out the drag rope which makes the dipper handle to nearly horizontal position. The bucket (which has lip and 4 to 6 teeth with wider opening area) tilts making it's face down to dump materials over the haulage unit.

Presently hydraulic back hoe are also available in the market. The capacity of the backhoe bucket may be in the range starting from 0.38 m^3 to over 18 m^3 (approx) , digging depth is around 4m to more than 8 m and the height of discharge is more or less half the boom length. It's efficiency is around 60% to 70% of that of a dipper shovel of identical size. As a safety measure the backhoe should be kept at a safe distance away from the crest of the bank to avoid danger arising due to caving. Although it has longer cycle time and less efficiency in discharging materials over the trucks but it can nicely be used for removing top soil and overburden. It is widely used for trenching and construction fields. Table 6.13 gives the brief specification of a few backhoe.

Fig. 6.12 : A Schematic Line Diagram of a Hydraulic Backhoe under Operation

Table 6.13 Backhoe hydraulic tripower shovel range (manufactured by O&K)

	RH30C	RH40C	RH75C	RH120C
Backhoe range (m^3)	1.8-4.7	2.0-5.6	3.6-10	6-18
Breakdown /crowd forces ,KN	240/250	330/330	415/450	700/750

Other details are given in the topic 'Hydraulic Shovel'.

A RH 40C backhoe shovel with trimatic attachment having average cycle time around 25 s can load limestone (sp. gravity 1.6) onto a 42 t dumper in 2 minutes 30 seconds and 16 dumper of that size can be loaded per hour producing around 672 t of ore per hour.

CHAPTER 7

TRANSPORTING EQUIPMENT

The selection of haulage equipment depends upon the following factors-

- Type, size and characteristics (sp. gravity, swell, stickiness, free flowing, etc.) of the overburden rock to be handled.
- Quantity of the material to be handled (that is the required production rate per hour, per day, per month or per year) and use of the haulage equipment for multiple purpose under variable operating condition.
- Condition of the surface terrain, plane or hilly terrain, altitude, etc,
- Gradient of haul,
- Mobility,
- Continuous or non-continuous operation,
- Maximum use of the equipment,
- Degree of compatibility,
- Various hurdles like cross over bridge, overhead clearances, good road condition and restrictions like width of the haul road, maneuvering space, side clearance, wet and weak road condition, etc.
- Climatic conditions, like air temperature (heat, cold) arid, rain, frost, etc. affect the selection,
- Length of haul, number of bends, radius of curvature, final and intermediate discharge points, etc.,
- Points of material discharge (e.g. crushing plant, waste pile, feeder bunker, etc.). Here size, capacity, type, etc. of the crushing plant, feeder bunker, etc. are very important,
- Type, size, capacity, cycle time of loading equipment like shovel, dragline, front end loader, bucket wheel excavator, surface continuous miners etc., are very important,
- Environmental condition like degree of noise, vibration, dustiness, fogginess, etc.,
- Availability of power,
- Accessibility in between the mining site to the dumping yard,
- Availability of capital and initial capital investment,
- Operating, maintenance and overall transporting cost,
- Life of the mine,
- Safety,
- Method of mining,
- Operating condition,
- Competency of surface soil or rock and their load bearing capacity,
- Indigenous availability of the machine and spare parts.

EXCAVATOR AND TRANSPORTING EQUIPMENT MATCHING

Excavators and transporting equipment should always be perfectly matched. It is already mentioned that an excavator is selected based on the targeted production, stripping method, pit geometry, diggability, operating condition, blending requirement, horizontal and vertical reach, maneuverability, compatibility, continuity of operation or discrete operation, surface topography, floor strength, physical characteristics of material (nature, type, size, etc.), flexibility in operation, climatic condition, machine reliability, safety, indigenous availability, capital and operating cost, etc. Based on the above condition, bucket capacity, machine characteristics (like positive crowd action, simple, strong and accurate spotting ability, mobility, sub- grade digging capability, maintenance and power consumption, cycle time, etc.), machine geometry (boom length for digging and dumping height and radius, bucket size, length, width and height of the main machine body, propelling mechanism, weight, safety items, stabilizing units and dimension of other accessories), etc. are selected. Pit geometry like average and maximum height of the mineral and overburden benches, bench slope angle, the width of the bench floor, spoil heap and the slope of the spoil heap, access room, etc., play the most vital role during selection of excavators. Space is an important criteria for the above selection since it directs the requirement of space for blast fragmented mass, loading and unloading range by a shovel, passage of trucks and other vehicles, etc.

Selection of the transport equipment should be such that the total system must match the production requirement. excavation or loading machine geometry like dumping height and radius. bucket capacity. machine

characteristics, etc. If trucks are selected , it would be better if truck is filled up by the shovel with only 3 to 5 passes which will load material evenly over the body of the truck and also lower down the cycle time and increase the production level. Less number of passes will lead huge amount of spillage where as more number of passes will increase the cycle time for loading. Truck cycle time like spotting, loading, maneuvering, hauling the loaded truck, spotting and dumping in the dump yard, empty return to the face, waiting, etc. are very important criteria for the truck selection. The cycle time should also include various delays in operation. Based on above ,the size of the trucks are selected. The parameters include the capacity, design, strength of the frame, body, canopy, motor mountings and other components, etc. and their reliability, capability and safety under static and dynamic loading condition.

Based on the physical dimensions of the truck the followings are selected-

- Economic flywheel power to gross machine weight (presently electric wheel transmission system, power more than 1800 kW and also up to 400 t capacity diesel electric trucks are available).
- Smoother, quicker and reliable power transmission system,
- Facility of ease of repair and maintenance.
- The width of the bench floor taking due consideration of flexibility of shunting and loading operation. One way or two way traffic system, safety of man and machine, etc.
- Number of truck requirements based on the cycle of operation, capacity of haul, etc.
- Width of the hauling road considering the factors like physical size of the trucks, density of the hauling trucks, minimum safety distance in between the two running trucks (one incoming and the other outgoing), distance of haul, minimum safety distance including footpath from both side edge of the road, etc. Table 7.1 shows the matching of shovel and dumpers.

Table 7.1 Matching of shovel and dumpers combination

Shovel with bucket capacity , m^3	Capacity of dumpers/trucks , t
0.9	8
2.5	25
3.2 to 5	35 to 50
5 to 10	50 to 77
10 to 15	77 to 120
15 to 20	120 to 170
20 to 35	170 to 200
35 to 45	200 and more

When continuous surface miner or bucket wheel excavator are to be employed and if continuity of operation is desirable, best proposition will be the deployment of either spreader-belt conveyor combination or BWE-mobile belt conveyor combination. The capacity of out bye conveyor circuit should always be more than the capacity of BWE or surface continuous miner. However the combination like BWE and rail is the cheapest among all provided the haul distance is more.

It should always be remembered that an excavator should never be idle. Hence to support an excavator for its maximum utilization time, the number of haulage units should always be more. In a series circuit if any unit in the system goes out of order, the total system will be stopped and there will be no production. Table 7.1 shows the matching of shovel and dumpers combination.

TRUCKS

Haulage trucks are one of the most important media of transport of coal, mineral or overburden rocks in the surface mining system. Capacity of dump trucks varies from 7 t to 400 t. Doubling of truck payload i.e. doubling of its productivity , will lower the unit cost of truck transport by up to 50%. Dumper of lower capacity is driven by diesel engine along with torque converters whereas higher power dumpers are mostly driven by diesel electric power trains. Higher capacity 320t and above capacity dumpers are moved by the A.C. electric drive. In some of the cases, the trolley assisted electric power dumpers are in use to protect the environment. Their speeds are also very high and the cost of transport is also very less since they use only electricity. A small capacity truck can negotiate a gradient of around 12° whereas a high capacity truck can negotiate a gradient around 7°, however the best gradient for any truck haulage system is less than 6°.

The radius of any good bends of haulage road are around 10m to 20 m. The chief merits of truck haulage system includes their simplicity in operation, ease of maintenance, flexibility, concentration of work, in built power arrangements, etc. The system can effectively be used for selective mining in deep mines to increase the output of shovel, to work in a mine with shorter life, in mining operation in the restricted area, in a mine which demands flexibility of production, in rough terrain, etc. The truck routs require high excavation works and hence cost of construction of roads are also very high. The system also requires large quantity of truck units, more drivers and helpers, causes high traffic density in the main haulage road, consume heavy amount of diesel oil and lubricants, operating and maintenance costs are heavy. They are also influenced by the condition of the roads and climate. To increase much of tyre life, the operating condition of the road should be very nice and hence it requires high investment in maintenance job. The economic travel distance by a dumper is around 3 - 4 km (best is within 2 km). Generally a truck is run at a speed ranging from 30 - 40 km/h. The speed and consumption of fuel depends upon the grade and condition of the roads, traffic density, maintainability, efficiency of the engine, amount of load, etc.

Out of available of various types, rear discharge dumper, bottom discharge dumper and side discharge dumpers are mainly used in the surface mining operation although in few cases trolley assisted electric powered dumper, road trains are also visible.

Few Salient Points for Selection of Transporting System

During selecting any transporting system in the surface mining system the factors to be taken into account carefully are as follows :

- Select shortest haulage rout taking due advantage of gravity (i.e. move the haulage unit towards the downward direction as far as practicable),
- Select one way haulage system when the capacity of haul is heavy otherwise two way system will be better on cost economic point of view,
- The main haulage road (tracks in case of locomotive) should be of permanent in nature. In the bench when tracks were installed they should be reinstalled after a sufficient lapse of time,
- Density and abrasivity of the loaded ore, mineral or rocks are to be found out carefully to ascertain the degree of impact thrust to be felt by a truck or railway wagon when a shovel, dragline or any other excavator load on them. Here the degree of wear also plays an important role,
- Maximum density of traffic allowable in a road for faster haul,
- Minimum number of bends in the haulage roads or tracks,
- Use of same type or minimum verities of transport system for better operation, maintenance and inventory control,
- The production capacity should match the excavator or loader capacity and the transport system capacity should match the capacity of the later,
- Transport system should be highly reliable, efficient and productive,
- Need of continuous and discrete operation, mobile or stationary, etc.
- Need of non conventional haulage system (e.g. skip, haulage, etc.) in place of conventional haulage system like trucks, railway, belt conveyor, etc.
- Safety of the system,
- Cost economics
- Climatic condition

Rear Discharge Dumper (Fig. 7.1 and 7.1(a))

In rear discharge dumpers soil, coal, blasted ore, or overburden rocks are loaded onto the main loading body attached at the rear side of the operator's cabin and are discharged through the rear end of the body by lifting the later with the help of two hydraulic rams of 10t to 15 t capacity. The outer cylinder end of the rams is attached with the chassis whereas the inner cylinder ends are connected with the main body. When the inner cylinder of the rams are retracted the main loading body again come to it's original position. Rear discharge dumpers are available in various design like 2 and 3 axle, 2 axle drive (it is common in 3 axle drive although some times they are also used in 2 axle

drive), dual tyre (used either in the single axle or in both the axle or in the rear 2 axle of the 3 axle dumper) etc. In case of large capacity dumper wider operating bench floor and turning radius are necessary and also their maneuverability is very poor. Rear discharge tractor trailer also comes under this category of the rear discharge dumper. In this case a trailer loading body is attached at the rear end of the tractor body. Load is discharged either manually or by hydraulic rams. The advantage gained in this type of system is that the trailer body is removable which permits the trailer free tractor to do some other auxiliary operation. It can nicely be steered and maneuvered and can use in the smaller width roadways. But their spotting during loading and dumping is very difficult. Rear discharge dumpers are very much useful in longer and steeper hauls due to their excellent traction facility.

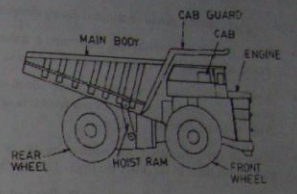


Fig. 7.1 : A Schematic Line Diagram of a Rear Discharge Dumper

Side Discharge Dumper (Fig. 7.2)

This type of dumpers are not much in use in the surface mining system for handling coal or other softer minerals. This type of trucks dumps the loaded materials in the side and also during plying by the side of the dump yard. In another version of side dump truck system e.g. in the side dump tractor trailer unit, there is an option of hoisting system either by an integral hydraulic arrangement or by an external ' sky hook' arrangement. Tandem trailer unit is attached with the tractor to balance it as well as to increase the capacity of payload.

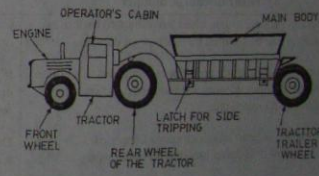


Fig. 7.2 : A Schematic Line Diagram of a Side Discharge Dumper

Bottom Discharge Dumper (Fig. 7.3)

This type of trucks (tare weight is low) are very much useful when the mineral or coal or rocks are to be handled are softer in nature and has free flowing characteristics. They are used when there is the existence of haulage road with milder gradient (below 5°) and there is the provision of windows over the dumping place in the dump yard. This type of trucks when handle coal is termed as 'coal hauler'. It is a very high speed vehicle. It is a diesel operated tractor unit coupled with a large trailer unit at the rear. The tractor unit has two smaller wheel in the front and two larger wheel (drive unit) at the rear, whereas the trailer unit contains two larger rear wheel. Out of many version of the bottom of the trailer unit, drop bottom type door is most common. It is actuated by the rope pulley system operated by a hydraulic cylinder. The main loading body of the trailer unit has slanting sides all around towards the bottom of it so that it has a wider surface area at the top compare to the surface area at the bottom to guide loaded materials to flow freely. Larger size ore or rocks should not be loaded over it otherwise it may cause jamming at the bottom opening. This type of trucks requires narrower turning radius. Table : 7.2 shows a comparison between rear and bottom discharge dumpers.

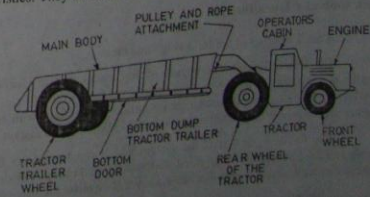


Fig. 7.3 : A Schematic Line Diagram of a Bottom Discharge Dumper

Safety Precautions during Dumping

The following precautions should be adopted during dumping operations by all types of dumpers:

- Where there is a possibility that the ground at a dumping place may fail to support the weight of a vehicle, loads should be dumped back from the edge of the bank;
- A vehicle should not be permitted to dump a load over a bank or into a bin, unless there is an effective back stop provided or otherwise (if necessary) a spotter is safely stationed and properly equipped to guide and direct the driver to a safe dumping position;
- When dumping is carried out, whether by day or by night, marker guides or other effective signs should be placed to indicate to the driver the limit of his safe approach to the tipping area;
- When dumping is carried out during the hours of darkness, the area should be illuminated by lighting units so placed as to give effective illumination to the working area and to the edge of the dump area and the mine operator or manager should ensure that adequate artificial lighting is provided at all points where poor visibility or work during the hours of darkness may cause a hazard.

Table : 7.2 A comparison between rear and bottom discharge dumpers

Rear discharge dumper	Bottom discharge dumper
1. Very large size, rough, blocky overburden rock or ore or mineral can be handled by this type of dumper.	Comparatively smaller size free flowing overburden rock, ore or mineral can be handled by this type of dumper.
2. It can negotiate more gradient as compared to a bottom discharge dumper. However it requires rapid spotting facilities in a limited area.	It yields good result in level gradient and up to an optimum gradient of 5°(say). However it can negotiate more inclination.
3. Dumping is generally done into a hopper of crusher plant or it dump onto the dumping yard.	Dumping is generally done into grizzlies placed on the level of the ground or into a hopper.
4. High degree of flexibility in operation is required for doing job under variable conditions.	Less flexibility in operation may be the sufficient to do the required job under variable conditions.
5. It suffers from severe shocks during loading and unloading.	Loading and unloading shocks are less in this case.

Main Units of a Truck/Dumper

A truck consists of the following main units-

The Power Engine-The engine of any trucks system should be of higher power and lower in weight. The specific power rating of a truck system varies from 6 - 8 kW/t of load. The engines (2 cycle or 4 cycle type) are operated mostly by diesel having supercharger or turbocharger along with inter cooling arrangements. Presently 16 cylinder engines with a power ranging of more than 1800 kW are in the market.

The Drive System-It is the system which supplies power from the engine to wheels. The system should be highly reliable, longer life, high rated performance and cheap. The drive system may be classified into many categories like (a) mechanical and hydro mechanical drive (b) hydrostatic drive and (c) electric drive.

Manually operated mechanical drive contains gear (around five) and couplings which are operated by air assisted clutches, gear operating system and steering mechanism. For operation smoothly in the downhill direction it contains retarder units which retard the movement of the vehicle during that operating condition. In some of the version a torque converter instead of coupling are employed. The manually operated drives are used for lower capacity trucks.

The larger size and the improved version trucks are all having hydro mechanical drive and there is always the presence of torque converter, hydraulic clutches, gear system and hydraulic transmission arrangement. It may have more

than 5 reduction gear ratio too. Road rangers are also introduced to lower down the weight of the drive. This type of drive consume more amount of fuel but driving is easy and gear changing is also very smooth. A lock up clutch is provided with the drive system to cut out the torque converter from the drive as well as to re-establish the mechanical continuity.

In electric drive system (Fig. 7.3(a)) drive wheel consists of a D.C. generator, an exciter, two (in a high capacity vehicle four in number) motorized wheels, a blower for cooling, etc. The traction motors, each of which is housed inside the rear wheel rim drives a series of sun and planetary reduction gear system which finally drive the wheel. Sometimes the D.C. traction motor is mounted ahead of the rear axle. The retardation and braking is achieved by the electric dynamic braking system; whereas emergency brake is provided for emergency stoppage of the trucks. Differential gearing system is also attached with the drive system. AC electric drive for higher capacity dumper is also available in the market. A basic differences between the mechanical and electric drive are pointed out in the table :7.3

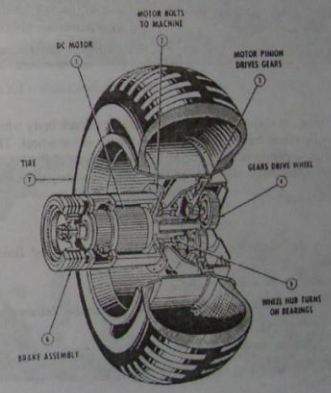


Fig. 7.3 (a) : An electric drive system

Table :7.3 Relative merits between electric drive vs mechanical drive trucks

Electric drive	Mechanical drive
• Average efficiency over a wider range of hauling condition is less as compared to an identical mechanical drive-truck. However maximum utilization of power is possible in full speed operation.	• Average efficiency over a wider range of hauling condition is more as compared to an identical electric drive-truck. However maximum utilization of power in full speed operation is not possible.
• Easier to handle both in uphill and downhill hauling. It has a better downhill speed control as compared to a mechanical driven truck.	• Difficult to handle both in uphill and downhill hauling.
• Only one speed reduction is possible.	• Choices of gearing ratio is possible.
• Retarder capacity is lower.	• Since the combination of both the retarder and engine establish the retarding power level, their retarding capacity is higher.
• Machine availability is more.	• Machine availability is less.
	• Mechanical components' weights are less and utilize a

• Weight of the electrical components are more.	higher percentage of engines gross power.
• Well matched to the profile of haul road.	• They are also equally matched to haul road profile.
• High capital cost.	• Capital cost is lower as compared to an identical electric drive truck.
• Very difficult to repair and maintenance. Highly skilled manpower is necessary for maintenance job.	• Very easy to repair and maintenance. Less skilled manpower are necessary for maintenance job.
• They are not much flexible in operation in level to steeper gradient.	• They are very much flexible in operation in level to steeper gradient by increasing or decreasing the speed accordingly.
• Less safe in hilly terrain since speed retarder capacity is lower.	• More safe in hilly terrain because of speed retardation.
• Power can be applied smoothly from standstill position to running condition, resulting less stresses on the drive transmission system.	• Power can not be applied smoothly from standstill position to running condition, resulting less stresses on the drive transmission system.
• Overall low maintenance cost.	• Overall higher maintenance cost.

Hydrostatic Drive

In this system a very large capacity pump is incorporated in the truck body which generates high pressure fluid (oil) and directed in the hydraulic prime mover located in each of the drive wheel. The motor can rotate in both the direction (clockwise and anticlockwise) depending upon the direction of flow of fluid in the prime mover. Because of the higher efficiency and limiting the speed range planetary gearing system is widely used with this system.

It offer the following main advantages-

- High reliability,
- As the load or working resistance increases or decreases, the supplied fluid pressure is also increases or decreases automatically,
- It helps easier control and operation of the machine.
- It also offer a step less variable speed control with the help of variable delivery pump and motor:
- It can nicely withstand shocks and vibration forces,
- Weight is lower as compared to the other drive system,
- It has lower fuel consumption,
- Maintenance cost is also less.

Fig. 7.1 (A) : Rear Discharge Dumper by Courtesy Jessop & Co. Ltd. India.

Fig. 7.4 : Rail Loading System by Courtesy Meco International, U.K.

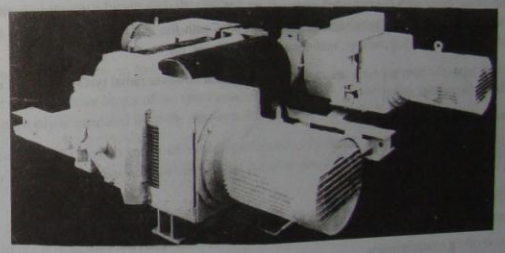


Fig. 7.10 : Belt Conveyor Drive by Country Meco International, U.K.

Torque Converter

In this case the blades are curved in both the impeller and runner units and a reaction member is added up with it. As the impeller rotates, the oil will also rotate and impinge into the stationary runner unit. From the runner unit oil moves reverse to the impeller rotation and directed against the stationary reaction member. Here the direction of rotation of oil is changed and it's energy is added up with the impeller. This will cause the torque in the runner unit which is equal to the impeller torque plus the reaction torque. Hence a torque converter automatically adjusts itself according to the requirement of load. If the input torque is constant the output torque of the runner unit increases as the speed decreases and attains at a level of its maximum ratio of torque multiplication at the zero speed. Torque converter may be divided into two types viz (a) pure torque converter and (b) torque converter coupling. In the former case the reaction member always remains stationary whereas in case of torque converter coupling the reaction member unit governs by converter range but rotates during the coupling range. The direction of rotation of the reaction member unit governs by the direction of impingement of the oil into its vanes produced by the runner unit. Hence converter coupling combines the advantages of both the fluid coupling and a hydraulic torque converter. A torque converter may be available either in single stage or in three stages and requires an outside cooling arrangement to lower down the heat produced by the converter within the range of 70°C however it can work at the temperature around 120°C.

Other Units

Suspension Unit :To save the frame and body presently hydro-pneumatic suspension system operated by nitrogen gas compressed by oil under pressure are widely used in heavy trucks for their high shock absorbing efficiency. Although in some low capacity trucks leaf spring suspensions are also used.

Hydraulic System: Hoisting and lowering operation as well as steering operations are done and controlled by the hydraulic system operated by a vane pump. The hydraulic rams for hoisting and lowering operations are performed by the two double acting rams. The ram is connected with the frame as well as with the body by ball and socket arrangements.

Body: The body of the off-highway trucks are generally of 'Standard' ,the 'V' type or modified' V' type made of thick, high strength, alloy steel. Sometimes aluminum alloy is also used for construction of the body to make it a lighter one. The standard type body has vertical sides and tapered from front to rear end, whereas the 'V' shape body has a constant angle floor plate sloping towards the front from the rear end of the truck having vertical sides. The body is also provided with a reinforced cab-guard (canopy) at the front to protect the operator's cabin. Sometimes wear plates, liner plates etc. are welded with the floor and side of the body respectively to resist severe abrasion and impact forces. The body may also be provided with rocker ejectors to clean the rock fastened in between the dual tyres.

Tyres :Both ply ratings and size of the tyres are selected based on the amount of load, gradient of the haulage road, heavy thrust forced on the body with the unloaded materials by a shovel, flotation and traction requirement, etc. The base of the tyres are selected based on the requirement of contact area with the surface, flotation, load to be carried out, reduction of vibration and tyre cutting damage, maintenance requirement, etc.

Out of many types the bias-ply tyres are most commonly used however radial tyres are also in use in many cases because of their following advantages-

- Ability to negotiate higher speed,
- Can carry more load ,
- Wear in the tread is also very less,
- Save much amount of fuel,
- On loose ground it has a good flotation ability,
- It has good riding characteristics.

Larger size tyres offers better tractive efforts and notation, offer well resistance to shock loads due to heavy and abnormal loading by the shovel or dragline and can negotiate high gradient with heavy load. Wide base tyres offers higher flotation and contact with the road, it has higher load carrying capacity, has lower maintenance cost and also gives good comfort condition to the driver. Dual tyres are seen almost in all the truck wheels particularly at the rear axle because of their higher stability and load bearing capacity. With dual tyre system longer haul at higher speed is possible.

The various types of tyres which are available in the world market are as follows:

- Tyre with tubes
- Tubeless
- Radial steel cord tyres
- Conventional cross - ply tyres
- Tyre with protective patterns and devices to prevent sudden failure
- Traction type tyres (Aggressive tread pattern)
- Rock type tyres (non-aggressive tread pattern)

- Deep treads ,
- Extra deep treads
- Conventional width tyre
- Wide base tyre
- Extra-wide base tyre (flotation consideration)

The tubeless tyres built-up lesser heat in the air and provides greater protection against blow-outs. Presently a chemical being introduced inside the tyre which shall seal off any leakage immediately in the event of latter is formed.

Tyre may be of thin walled (for the purpose of ordinary transport vehicle) or it may be of thick-walled (for the purpose of use in heavy vehicles like front-end-loader, dozer, heavy dumpers, etc.). Overtiring shall be done in case of a vehicle has to be run over a longer distance.

Incase of tubeless tyre, the tube becomes an integral part of the tyre casing and flaps are not required in this case. Tubeless tyres offer the following merits:

- Eliminates the tube and flap replacement cost.
- Assists cooler running of the tyre in hot climatic condition and longer haul distance.
- Bursting phenomenon is reduced. It is safer, leakage if occurs in it is very slow and before completely damaging of the total tyre system efficient repairing can be made in time.
- Very easy and faster in fitting with the dumper.
- Considerable reduction in down time since tube and flap are eliminated.

Some of the important definitions on tyres are as following

Plies : These are the successive layers of the tyre casings. Each cord in a ply is completely surrounded by resilient rubber compounds and each ply is insulated from the next one. Ply rating is the index of the tyre strength.

Casing: The casing of the tyre is made of rubber cushioned cord or fabric (nylon, perlon, polyester, etc). It is the backbone of any tyre which assists confinement of the inflation pressure to support the external load and cushions the shock load.

Breakers : The breakers means the extra plies located under the tread which helps to absorb sudden shock and gives extra resistance to the tyre against blows received from rough surfaces or obstacles.

Tread : The tread of the tyre contacts the road surface. It assists efficient traction and offers resistance to cuts and wear.

Beads: The beads of the tyre anchor the latter to the rim. All plies are tied up into these bundles of strong wire which prevent any change of shape or fit to the rim.

Sidewalls: On both the outer sides of the casing there are the protective rubber coverings which are termed as the sidewalls. The sidewalls are designed to flex and bend without cracking when subjected to continuous deflection or tremendous sudden shocks.

Tube and Flap: The tube retains air under pressure within the casing of a tube type tyre. The flap protects the tube from injury by the rim and tyre beads.

BIAS PLY

The bias-ply tyre is made of layers of rubber-coated plies composed of textile cords, usually nylon, placed upon each other at approximately 30-degree angles. These plies are then wrapped around the bead wires which anchor the tyre to the rim to form the casing, or air chamber. The plies are thereafter covered with more rubber to form the tread pattern.

The only way to increase the strength of bias-ply tyres is to increase the number of plies and bead wires. More plies means more mass, increasing heat retention, reducing tyre life. The bias-ply tyre casing is constructed to form one

working unit. When the sidewalls deflect, the tread squeezes in and distorts. This makes the tyre scrub or squirm, decreasing traction, operator control and accelerating tread wear. Since the tread is subjected to any flexing that occurs, the footprint or contact patch distorts. This distortion causes abrasion from the ground surface, which reduces the life of the tyre. All of these factors significantly increase fuel consumption. Bias-ply tyres have reduced life due to fast wear, high fuel consumption and reduced grip and control.

RADIAL

It is a pneumatic tyre in which the ply cords extending to beads are laid at approximately right angles to the center line of the tread. The radial tyre is constructed in two parts. First, a single layer of rubber-coated steel cables arch from one bead to the other to form the tyre casing. Second, numerous rubber-coated steel belts are placed in the crown, under the tread, to form a strong stabilizing unit.

To increase a radial tyre's strength, larger diameter steel cables are used. These steel cables help reduce punctures, tears and flats. They also help distribute heat, resulting in a cooler running tyre, improving fuel economy. The combination of steel stabilizing belts and the single-layer radial casing allows the tread and sidewall to act independently. The sidewall flexes easily under the weight of the vehicle and its cargo, while the tank-track type tread provides even contact with the ground. Greater vertical deflection is achieved with radial tyres. This is desirable because extreme flexing greatly increases resistance to punctures while protecting the operator and expensive vehicle components. The stabilizing steel belts and radial construction minimize tread distortion. As the sidewalls deflect, the belts hold the tread firmly on the ground, minimizing tread scrub and greatly increasing tread life. When negotiating curves and encountering side forces, the independent action of the tread and sidewalls keeps the tread flat on the road. This allows the tyre to hold to its path. Independent road tests showed the radials made lane changes on wet pavement 10 percent faster; stopped in a 21 percent short distance; ran 27 percent cooler and negotiated curves 15 percent faster. These proven performance results are why trucks and automobiles use radial tyres.

Right types of tyres to be used

The wide base tyres are presently being in use in heavy load condition and also in softer rough ground condition. The wide base tyre offers a wider tread, a greater volume and consequently can be operated at the lower inflation pressure, compare to the equivalent normal sectional tyre for the same load condition.

Deep treaded tyres are designed for using in front-end -loaders, dozers, scrapers, etc. where haul routes are quite short and cycle of frequency of operation is low. The tyre has the premium pattern depth with a tougher rubber compound.

Because of the higher amount of rubber in the tread pattern, it offers the lower operating cost. The following types of tyres are the best suited for the following machineries:

1) Rear dumper: For smaller-haul distance use deep tread tyre whereas for longer haul distance use standard tyre.
2) Bottom dump-coal hauler: Twin tyres are used in place of single wide base tyre to avoid production of high temperature in longer haul distance.
3) Scraper : Use wide base deep treaded tyre
4) Front-end-loader: Use side base deep treaded tyre since it has low centre of gravity and high vehicle stability.
5) Dozer :Use wide base deep treaded tyre.

Tandem Drive

The carrying capacity of the dumper is increased by putting another axle at the rear. Both the rear axle is the driving axles. This is called the tandem drive. The number of wheels may be 10. The tandem drive also improves traction and minimizes the driving strain. The propeller shaft crosses the top of both the differential gearing system of both the rear axles, drives them either through worm gearing or hypoid with double reduction gears. The shaft between the 2 wheel axles is fitted with a pair of universal joints and a sliding coupling.

Differential Gearing System

In case of 2 speed gearing system, it works through the sun gear set whereas in case of high speed range, the ring gear turn the carrier and spider gears directly. The smaller gears turn freely on the cross-shaft which is connected to

the ring gear. A 2 sided clutch jaw is splined to the centre of the shaft which is controlled by a vacuum diaphragm located on the outside of the housing. When the teeth of this gear locks with the teeth of any of the gear, power is transmitted to that gear and through the meshing large gear to the differential gear. The live axles are splined into the axle gear in the differential and are bolted to the hubs, so that they revolve on the bearings supporting these parts. The hub takes the full drive torque of the dumper and transmits power to the wheel through studs and friction contact.

Vacuum Brakes

It consists of a vacuum cylinder and a pusher ram. The atmospheric air pressure pushes the piston into the vacuum on the other side of the piston chamber and brake is put in 'ON' position for debarking the reverse action, is done by a spring making both the side vacuum. This vacuum cylinder acts as a booster to the braking system. The booster contains a vacuum chamber, a slave chamber and a 4-chambered control valve. The booster reduces the required pedal pressure from 25% to 75%.

Clutch

The friction (plate type) clutches are used in the engine flywheel. The clutch is engaged by springs and released by operating a foot pedal, situated at the operator's cab.

Operators Cab

The cab is made of sheet steel and glass and is of self supporting type. The driver seat is well designed. All clutch, brakes and accelerator pedals, steering, gearshift levers, speedometer, etc. are located in the driver's cab. Various safety items including driver seat belt, mirrors, etc. are also fitted into it. Higher capacity dumpers cabs are totally air conditioned. Most of the dumpers contain power steering which consists of a two-way ram with the built-in valve system and is powered by a hydraulic pump operated by an engine.

Causes of Tyre Damage and Their Remedies

The following are the main causes for damaging and decreasing the tyre life and their remedies :

- If the tyre is inflated with high pressure then it's bead will be shaped into a round one and only the centre portion of the tyre bead will come under contact with the road. Because of this reason the centre portion will come under heavy load and the thickness of the bead decreases due to wear. On the other hand if the pressure inside the tyre is low, the centre portion of the bead will be deflected inside and both the outer edges will come under severe load condition and will be worn out. Hence tyre must be inflated at an optimum pressure keeping due regard of the designed pressure, load over the tyre, temperature of the surrounding atmosphere and also the temperature produced by the frictional resistance. High pressure inside the tyre also causes heavy stresses on the cords causing tyre puncture. Under inflated or over inflated tyres bends back and forth causing internal shear ,damaging of plies and production of heat which reduce the strength of the rubber and plies and ultimately they may fail if the rise of temperature is heavy.
- If the roads are not smooth spillage and other small objects or metal pieces scattered all over the road, these objects come into contact with the tyre bead, causing hammering action onto it. This will also lead severe wear and damage of the tyre bead. Sometimes even when these objects are caught in between the tyres , they damage the sidewall of the later. Cutting of rubber is more severe during rainy season. Motor grader may efficiently be used to clean the spillage materials. Spillage from the truck can be controlled if tail gate is fitted behind the trucks enhancing its struck capacity. However it will increase the time for discharging material and hence the cycle time. During rain the trucks should not be run over the road to protect the surface of the later. The top of the surface of the road better be constructed with 50 cm to 80 cm asphalted concrete which will reduce both the tyre and fuel cost.
- Improper and careless assembling of tyre will cause damaging the contact area of the rim. So fitting of the tyre should be done carefully. If axles are not aligned properly, there will be wear inside or outside edge of the tyre depending upon the sagging of axle or outward leaning of wheel.
- Damaging of shock absorber, bead, etc. causes much of wear and damaging of the tyre.

- If the tyres are loaded heavily, there will be severe wear on the tyre because of under inflation. So avoid overloading.
- Speed of the wheel should be as per the designed one. If the speed is higher compare to the speed for which they are designed then huge amount of heat will be generated which can not be dissipated in the faster manner and will cause the damage of the tyre. Physical damage on the tyre due to the impact effects by the objects is higher with high speed.
- If more number of braking is done during running, there will be increase in tread wear and ultimately tread may comes off the tyre.
- If the speeds of the trucks are heavy and if the wheels of the trucks are not aligned, severe cupping (alternating hollows and ridges) may be formed on tyres. So speed should be moderate.
- Retreading (by buffing the worn out tread up to the required depth near to a good ply, vulcanization to be done to reshape it just like an original tyre) or recapping (replacing only the tread) of tyres for light to medium duty vehicles which run at a slower speed over smooth soil having less amount of rock will definitely increase it's life to a considerable amount. Retreading of tyre should be done in time.
- Hot climatic condition reduces the tyre efficiency since it has a poor capacity to disperse heat; whereas in cold climatic condition there may be the problems of relieving stresses from the steel attached with the tyre system. A sound road condition will solve these problems a little.
- If tyres are come in contact with grease, oil or other lubricant, the tyre will become soft and hence its life will be impaired. So tyre should not come in contact with any oil or grease.
- If dual tyres are used they should be of same size, equally worn out condition, same ply ratings and inflated to equal pressure so that both of them share the equal load and will prevent high wear to a particular tyre.
- Inspect regularly the condition of tyres like wear, cuts, imbedded rocks or metallic substances, rock particles in between the dual tyres, etc. Clean the foreign materials and take proper action to rectify the defects due to wear, cuts, etc. otherwise the condition of tyre will be further deteriorated.
- Efficient maintenance of both the tyre and the haulage road will increase the tyre life.

Safety Devices in Dumpers

The following safety devices should be incorporated in a dumper :

- Emergency steering switch with warning light to be used during emergency when the red light is on. It means that emergency steering system is in use.
- Twin-air horn for front and back-up warning.
- Air over hydraulic service brakes with emergency relays. Disc type brakes should be used in place of shoe brake. All dumpers should be provided with adequate braking systems for effectively stopping and holding the vehicle stationary when fully loaded and under any conditions .
- Parking brake with warning light for parking and to avoid backward sliding while climbing up a grade when the light will be on. It will mean that the parking brake is not released.
- Service and parking brake interlocks.
- Dynamic retarding with continuously rated retarding grids.
- Rear lights including two amber (for dynamic brakes), two red (for service brakes) and two white (for back up) are to be provided. An audio visual alarm with solid state media will be found the most suitable in the coming near future.
- Full time power steering system.
- Emergency engine shut off to stop the engine in case of emergency.
- Radiator fan guards.
- Tipping body props or other devices .These should be capable of preventing a skip collapsing and their mode of operation should be independent of the tipping mechanism.

- Wheel brake lock and drive system interlock.
- Key switch - it is a locking device that prevents machine use by unauthorized person. Key switch must be turned on to complete the circuit to the starter control.
- Circuit breakers operated at 24 volts.
- Seat belts for the driver.
- Hoist interlock. An indicator must be incorporated with the dumper for showing the position of the bucket during dumping and raising.
- Body-up buzzer. Buzzer warns against higher coolant temperature
- A good communication system must be incorporated with a dumper to visualize and taking decision for siding the same while any car or jeep is allowed to be passed.
- Engine oil pressure gauge to indicate the pressure of the engine's lubricating system generally incorporated in 25t - 35 t dumper. The pressure varies from 2.7 kg/cm^2 to 5.23 kg/cm^2 and 0.35 kg/cm^2 to 1.05 kg/cm^2 when the machine is in operation and idle respectively.
- Engine temperature gauge to indicate the temperature of the coolant in the engines cooling system. Under normal operating condition when the engine warms up, the engine temperature varies from 74°C to 85° C for a dumper of capacity 25t to 35 t.
- Air pressure gauge to indicate the amount of compressed air pressure in the main reservoir. A pressure around 8.2 kg/cm^2 is generally shown for a dumper of capacity 25t to 35 t.
- An automatic reverse signal alarm which is audible above the surrounding noise level.
- A warning buzzer will indicate the pressures below the safe operating range.
- Transmission oil pressure gauge to indicate the pressure of the hydraulic fluid.
- Transmission temperature gauge to indicate the temperature of the hydraulic fluid. Under normal operating conditions the temperature range varies from 71°C to 99°C while the torque brake is off in case of a dumper of capacity ranging from 25t to 35 t. The temperature should not exceed 121°C.
- Road condition control valve to allow the operator to select a braking action to compensate for 'dry' or 'slippery' road conditions.
- The front/wind shield glass should be of non-breakable type. The operator's cabin should be well designed, substantially built to ensure adequate protection to the operator against heat, dust, noise, etc. and also to provide adequate safety to operator in the event of overturning of heavy earth moving machinery.
- All the dumpers above 85 t capacity or more must be controlled by the microprocessor based control system. To protect the engine from fire, the following items are to be provided:
  1. Baffle plate between the turbo-charger and exhaust pipe.
  2. Trip relays in all the circuits except the battery circuit to detect short circuiting.
  3. The design of hydraulic pipe and exhaust route should be properly oil sealed to prevent spillage of oil on the exhaust route.
  4. Automatic fire fighting equipment including alarm should be provided.
  5. Asbestos coverings are to be provided on the pipes and manifolds which do not carry exhaust. It is to be provided to prevent fire if any oil dripped in them.
  6. A group of audio visual alarms.
  7. The driver's cabin of the high capacity dumper/truck should be air conditioned.
  8. The routine condition monitoring (RCM) arrangements should be provided in higher capacity (80 t and above) dumpers. The RCM system incorporated in the dumper provides the following facilities-
     - Visual inspections of all electrical and mechanical parts or units where the sensors are kept.
     - Vibration, ventilation and temperature analysis and study of all the critical parts and components of the dumper/trucks.
     - Sampling of hydraulic and gear oils and also checking of lubrication system.
     - Communication tracking in D.C. machines.
     - Checking of all electrical parameters.
     - Observation and monitoring of all general factors those effect safety of the dumper.

Standard Equipment Attached with a Dumper

The following standard equipment are attached with a dumper :

Ammeter, twin air horns, windshield wiper, air pressure gauge, air cleaner restriction gauge, engine oil pressure gauge, engine temperature gauge, transmission oil pressure gauge, transmission oil temperature gauge, tacho-hour meter, Speedo-meter road condition valve, wheel brake lock, starter key switch, head and tail lights, back-up lights, emergency power steering, outside right hand mirror, transmission shift indicator, head light high beam indicator, standard tool kit etc.

If needed the following optional items may also be incorporated: cold weather starting aid, hydro-air service kit, outside left hand mirror (for seeing view), rock ejector, etc.

Various other Safety Aspects

- No dumper /truck should be left unattended unless (a) the controls are in the neutral position and the parking brakes fully applied (b) if it is parked on a grade (c) the master switch is in the off position (d) all operating controls are in the neutral position, and (e) the brakes are set or other equivalent precautions are taken against rolling.
- When a dumper /truck is in motion between work areas, the movable parts of the equipment should be secured in the travel position.
- The engine exhaust gases from dumper /truck operated by an internal combustion engine should be discharged at a point remote from the operator's position and from any air-conditioning intake where fitted.
- The operator's cab of any vehicle used for rock transport should be so constructed or reinforced as to resist damage by spillage. The operator's cab should not be additionally equipped, altered or otherwise modified in a manner which impairs operating visibility. Cab windows should be of safety glass or equivalent, in good condition and should be kept clean. Cabs of mobile equipment should be kept free of extraneous materials. The design and layout of operators' positions and controls should conform with best ergonomic principles. Particular attention should be paid to layout and direction of movement of all control devices ,the operator's seating, protection against environmental hazards including inclement weather, heat, cold, noise and airborne dust and safe entry and exit for the operator.
- An effective audible warning signal should be sounded whenever the driver wishes to move the vehicle without having clear vision immediately in front and behind.
- Where appropriate, passing places should be provided on single-lane roads which should be clearly visible from both directions. The gradient and radius of any part of a road should be such that vehicles can negotiate the road with safety.
- The manager should arrange such signs as may be necessary to control the speed and movement of all dumpers /trucks making use of the roads.
- All low points on mine roads should be adequately drained so as to eliminate accumulations of standing water.
- Haulage roads with planned gradients in excess of 5 per cent should have emergency escape roads, which should be spaced throughout the length of the haulage road and ensure that a runaway vehicle entering an emergency escape road can be safely brought to rest.
- The operating speed of dumper /truck should be consistent with conditions of roadways, grades, clearances, visibility, traffic and the type of equipment used.
- All operators should maintain full control of the dumper /truck while it is on motion. Haulage equipment should be operated under power control at all times.
- Where loads are carried which project beyond the sides of dumpers /trucks or by more than 1.2 m beyond the rear of the dumpers /trucks, there shall be attached at the end of the projection a warning light during the hours of darkness or in limited visibility and a clearly visible warning flag of suitable size in daylight conditions. Where overhead clearance is restricted, warning devices should be installed and the restricted area should be conspicuously marked.
- All ramps and dumping facilities should be of substantial construction and have suitable width, clearance and headroom to accommodate the equipment using the facilities.

- Berms, bumper blocks, safety hooks or similar means should be provided and maintained as required to prevent over travel and overturning at all tipping and dumping locations.
- Chute loading installations should be so designed and installed that the persons operating them are not required to be in a hazardous position whilst at work. Any attempt to free a blockage in a chute should only be made by a competent person who fully understands the hazards involved. All grizzlies, grates and similar stationary sizing devices should be securely mounted and anchored. Persons required to work on a grizzly while breaking rocks should be provided with and should wear a securely anchored safety belt.

Table 7.4 and 7.5 shows the specification of various types of dumpers whereas the Table 7.6 shows the specification of off-highway truck used in surface coal mines in India.

Table 7.4 Specification of various types of dumpers

Manufacturer	Model	Nominal pay load (t)	Gross engine power (kW)	Transmission system	Wheels (Number x Size)
Caterpillar	785 789	150 195	1030 1343	- -	- -
Webco	190 3200	172 214	1340 1850	Electric(3)* Electric(4)*	6.36"x51" 10. 36"x51"
Euclid	R170	155	1190	Electric(3)*	6,36"x51"
Unit Rig	MK36	155	1190	Electric(3)*	6,36"x51"
Komatsu	HD1600M	160	1190	Mechanical	6,36"x51"
Titan	33-15C	172	1340	Electric(4)*	6,36"x51"
Wiseda	KL2450	200	1340	Electric(3)*	6,40"x57"

Note : * Electric wheel motor, locomotive type electric traction motor, Moderate life varies from 5000 to 10000 hours.

Table : 7.5 Specification of various types of dumpers

Manufacturer : UNIT RIG		
Type	Drive	Pay load ,(t)
Lectra Haul Mark 24	Diesel electric	90.7
Lectra Haul Mark 30	Diesel electric	108.8
Lectra Haul Mark 33	Diesel electric	136.0
Lectra Haul Mark 36	Diesel electric	154.2
Lectra Haul MT 4000	Diesel electric	217.7
Lectra Haul BD 240	Diesel electric	217.7
Lectra Haul BD 270	Diesel electric	244.9
Manufacturer : CATERPILLAR Model 785 , Flywheel power: 963kW , Gross power :1030kW , 150t payload Model 789 , Flywheel power :1271kW , Gross power: 1343kW , 195t payload		

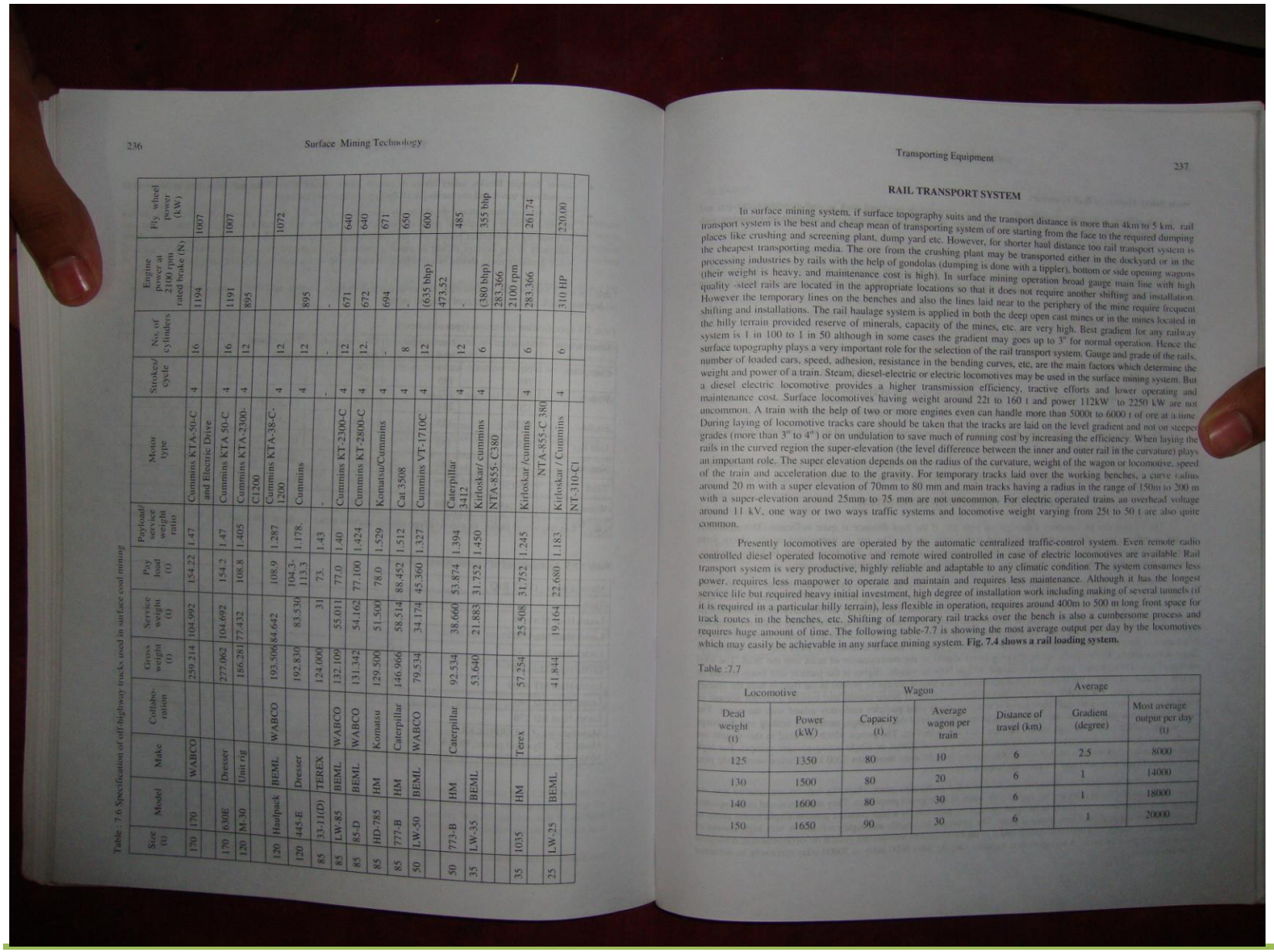


Table : 7.6 Specification of off-highway trucks used in surface coal mining

Size (t)	Model	Make	Collabo-ration	Gross weight (t)	Service weight (t)	Pay load (t)	Payload/ service weight ratio	Motor type	Stroke/ cycle	No. of cylinders	Engine power at 2100 rpm rated brake (N)	Fly wheel power (kW)
170	170	WABCO		259.214	104.992	154.22	1.47	Cummins KTA-50-C and Electric Drive	4	16	1194	1007
170	630E	Dresser		277.062	104.692	154.2	1.47	Cummins KTA 50-C	4	16	1191	1007
120	M-30	Unit rig		186.281	77.432	108.8	1.405	Cummins KTA-2300-C1200	4	12	895	
120	Haulpack	BEML	WABCO	193.506	84.642	108.9	1.287	Cummins KTA-38-C-1200	4	12		1072
120	445-E	Dresser		192.830	83.530	104.3-113.3	1.178.	Cummins	4	12	895	
85	33-11(D)	TEREX		124.000	31	73.	1.43	-	-	-	-	
85	LW-85	BEML	WABCO	132.109	55.011	77.0	1.40	Cummins KT-2300-C	4	12	671	640
85	85-D	BEML	WABCO	131.342	54.162	77.100	1.424	Cummins KT-2800-C	4	12.	672	640
85	HD-785	HM	Komatsu	129.500	51.500	78.0	1.529	Komatsu/Cummins	4	-	694	671
85	777-B	HM	Caterpillar	146.966	58.514	88.452	1.512	Cat 3508	4	8	-	650
50	LW-50	BEML	WABCO	79.534	34.174	45.360	1.327	Cummins VT-1710C	4	12	(635 bhp)	600
50	773-B	HM	Caterpillar	92.534	38.660	53.874	1.394	Caterpillar 3412	4	12	473.52 -	485
35	LW-35	BEML		53.640	21.883	31.752	1.450	Kirloskar/ cummins NTA-855- C380	4	6	(380 bhp) 283.366	355 bhp
35	1035	HM	Terex	57.354	25.508	31.752	1.245	Kirloskar /cummins	4	6	2100 rpm 283.366	261.74
25	LW-25	BEML		41.844	19.164	22.680	1.183.	NTA-855-C 380 Kirloskar / Cummins NT-310-CI	4	6	310 HP	220.00

RAIL TRANSPORT SYSTEM

In surface mining system, if surface topography suits and the transport distance is more than 4km to 5 km, rail transport system is the best and cheap mean of transporting system of ore starting from the face to the required dumping places like crushing and screening plant, dump yard etc. However, for shorter haul distance too rail transport system is the cheapest transporting media. The ore from the crushing plant may be transported either in the dockyard or in the processing industries by rails with the help of gondolas (dumping is done with a tippler), bottom or side opening wagons (their weight is heavy, and maintenance cost is high). In surface mining operation broad gauge main line with high quality -steel rails are located in the appropriate locations so that it does not require another shifting and installation. However the temporary lines on the benches and also the lines laid near to the periphery of the mine require frequent shifting and installations. The rail haulage system is applied in both the deep open cast mines or in the mines located in the hilly terrain provided reserve of minerals, capacity of the mines, etc. are very high. Best gradient for any railway system is 1 in 100 to 1 in 50 although in some cases the gradient may goes up to 3° for normal operation. Hence the surface topography plays a very important role for the selection of the rail transport system. Gauge and grade of the rails, number of loaded cars, speed, adhesion, resistance in the bending curves, etc, are the main factors which determine the weight and power of a train. Steam, diesel-electric or electric locomotives may be used in the surface mining system. But a diesel electric locomotive provides a higher transmission efficiency, tractive efforts and lower operating and maintenance cost. Surface locomotives having weight around 22t to 160 t and power 112kW to 2250 kW are not uncommon. A train with the help of two or more engines even can handle more than 5000t to 6000 t of ore at a time. During laying of locomotive tracks care should be taken that the tracks are laid on the level gradient and not on steeper grades (more than 3° to 4°) or on undulation to save much of running cost by increasing the efficiency. When laying the rails in the curved region the super-elevation (the level difference between the inner and outer rail in the curvature) plays an important role. The super elevation depends on the radius of the curvature, weight of the wagon or locomotive, speed of the train and acceleration due to the gravity. For temporary tracks laid over the working benches, a curve radius around 20 m with a super elevation of 70mm to 80 mm and main tracks having a radius in the range of 150m to 200 m with a super-elevation around 25mm to 75 mm are not uncommon. For electric operated trains an overhead voltage around 11 kV, one way or two ways traffic systems and locomotive weight varying from 25t to 50 t are also quite common.

Presently locomotives are operated by the automatic centralized traffic-control system. Even remote radio controlled diesel operated locomotive and remote wired controlled in case of electric locomotives are available. Rail transport system is very productive, highly reliable and adaptable to any climatic condition. The system consumes less power, requires less manpower to operate and maintain and requires less maintenance. Although it has the longest service life but required heavy initial investment, high degree of installation work including making of several tunnels (if it is required in a particular hilly terrain), less flexible in operation, requires around 400m to 500 m long front space for track routes in the benches, etc. Shifting of temporary rail tracks over the bench is also a cumbersome process and requires huge amount of time. The following table-7.7 is showing the most average output per day by the locomotives which may easily be achievable in any surface mining system. **Fig. 7.4 shows a rail loading system.**

Table :7.7

Locomotive		Wagon		Average		
Dead weight (t)	Power (kW)	Capacity (t)	Average wagon per train	Distance of travel (km)	Gradient (degree)	Most average output per day (t)
125	1350	80	10	6	2.5	8000
130	1500	80	20	6	1	14000
140	1600	80	30	6	1	18000
150	1650	90	30	6	1	20000

Some Safety Matters in Rail Transport

- Road-beds, rails, joints, switches, frogs and other elements of every rail track as well as bridges, culverts and other structures for supporting should be designed, installed and maintained in a safe manner consistent with the speed and type of haulage.
- Every train should be equipped with an effective braking system.
- Every locomotive, rail car, truck, wagon or other rail track vehicle should be provided with effective brakes which should be capable of being operated individually by hand and by the train braking system when forming part of a train.
- Positive-acting stop blocks, derail devices or other adequate means should be installed wherever necessary to protect persons from runaway or moving railroad equipment.
- Every locomotive should be equipped with the effective headlights and rear-lights; minimum two braking systems (hand brakes and pneumatic or electric), a whistle or siren capable of giving clear and distinct warnings and signals; sanding devices, speedometers, adequate fire extinguishers and first-aid equipment.
- Rocker-bottom and bottom-dump railcars should be equipped with locking devices.
- Where necessary, bumper blocks, buffers or the equivalent should be provided at track dead-ends.
- Track guard-rails, lead rails, frogs and guide rails should be protected or blocked so as to prevent a person's foot from becoming wedged.
- Public and permanent railroad crossings should be posted with warning signs or signals, or should be guarded when trains are passing and should be planked or otherwise be filled in between the rails.

PIPE LINE TRANSPORTATION SYSTEM

Ore or coal can be transported through the pipe line if the haul distance is quite sufficient. This system is selected in many cases since it is a quite cheap transporting media. The ore is transported in the slurry form (homogeneous mixture is the best as compared to a heterogeneous mixture). The total system includes crushing, screening, dewatering, secondary and tertiary crushing along with screening and mixing with water.

The slurry is transported through the pipe line (diameter ranging from 15 cm to more than 60 cm) of length varying from 5 km to more than 1500 km (economic limit) with a capacity ranging from 0.2 Million Tonne to more than 10 Million Tonne per year depending upon the types of ore or mineral. The efficiency of the slurry movement through the pipe lines depends upon the size (range varying from 100 mesh to -2 mm), shape, specific gravity of the ore particles, concentration of the solids in the fluid, velocity of the slurry (range varying from 1.2 m/s to around 2 m/s) and also the distribution of the particle size in the slurry. A specific gravity of ore or mineral around ±1 to more than 5 is suitable for slurry transportation. Higher is the specific gravity lower should be the concentration of solid into the fluid. The total amount of solid concentration in the fluid is around 40% to 60% by weight. Higher is the particle size lower should be the concentration of solid in the fluid. A certain amount of fines should always be present in the fluid to hold the coarser particles in the water. The particle when precipitated due to closure of pump should also contain a certain amount of water around its periphery so that when pump is restarted again solid particles uniformly mixed with the water. The system varies according to the number and type of pumps, diameter of the pipe range, number of feeding and discharge stations, the surface profile through which the pipe lines are to be laid. The pumps which are used in this system are mainly positive displacement type (generally double acting piston pumps or single acting plunger pumps). For high discharge capacity, power of a double acting piston pump may be even more than 1200 kW. Carbon-steel made pipe lines with sufficient wall thickness are generally used in this system.

Ore, coal, etc. from the stockpile are feed into the preparation plant where the formers are crushed into the required sizes to suit the system and are mixed with water in appropriate proportion. They are then stored in the agitated storage tanks from where the slurry is pumped through the pipe line by a high pressure positive displacement pump to transport to the required receiving terminal plant. The system is less costly and highly reliable. A very high amount of material can be transported with this system and it is the best mode of transporting system on environmental, ecological and a esthetic point of view. A transportation capacity varying from 6000 t/day to 30000 t/day can easily be achieved with this system.

An Example

Kudremukh iron ore mines of India introduced a 67 km long slurry pipe line for the transportation of concentrate from Kudremukh to Mangalore. The entire operation from mine to port is fully integrated and computer controlled.

Run-of-mine ore is dumped into gyratory crushers of capacity 4000 t/h . There are three gyratory crushers located at different locations at 1015m, 880m and 840m reduced levels. After crushing, the ore is transported to plant via 0. 35 Mt monsoon stockpile and through 6000 t/h capacity conveyor belts, where it is stocked into 8 silos of total capacity of 72,000 t. The transportation system from crushers to silos is fully integrated with the main computer situated in the concentrator plant. The ore, stocked in silos is fed to 4 autogenously mills at an average feed rate of 1000 t/ h for each mills where ore is crushed to the size of -20 mesh. Further, it is fed to 12 ball mills to reduce the particle size to 60% of -325 mesh. Magnetic iron particles are separated in magnetic separation circuit in primary, secondary and tertiary stages. Silica is further reduced in 8 nos. of column flotation. Non- magnetic particles are separated in non-magnetic circuit with 2 mechanical flotation cells. Recovery percentage is about 85% in magnetic separation circuit and 15% in non-magnetic separation circuit. Finished product is taken into concentrate thickeners and tailings which constitute about 65% of the ROM, are taken to tailing thickeners. Tailings are pumped to Lakya dam which is having a height of 100 m and a storage capacity of 245 million cubic meters. This is done effectively to prevent tailings going to permanent water sources to prevent water pollution. Finished product in terms of concentrate are pumped to Mangalore by 67 km long hydraulic transport system (pipeline). For about 13 km the slurry is pumped to the highest peak and the remaining 54 km is transported by gravity.

The followings are the brief data of Kudremukh iron ore project-

Length of the slurry transportation pipe line :	67 km
Diameter of the slurry pipes :	406 mm to 457 mm
Capacity of stockpile -I	0.3 MT
Capacity of stockpile - II	0.3 MT
Monsoon stockpile :	0.5 MT
Crusher capacity :	4000 t/h
Silos (fine) :	15000 t (- 127 mm size)
Silos (coarse) :	8000 t (+ 127 mm size)
Export quality Fe-content:	Average : 66.5%, Minimum : 64%
Silica and aluminum content:	Average : 4%, Maximum : 5%
Size:	100% below 100 mesh 60% below 325 mesh
Moisture content (average) :	9%, Maximum : 10%

The flow diagram of iron ore tranportation system through the pipe lines is shown in the fig. 7.4 (b)

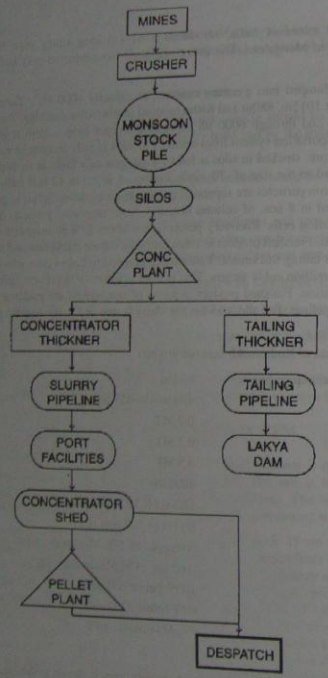


Fig. 7.4 (b) Shows the flow diagram of pipe line transporting system of Kudremukh iron ore project of India.

BELT CONVEYOR SYSTEM

Belt conveyor system is one of the most vital transporting system both in underground as well as in the surface mining system. They are available both in mobile (high angle belt conveyor presently used in the face of mechanized surface mining system) and in stationary form. Belt conveyors systems are widely in use for bulk material handling system for coal, lignite, gypsum, bauxite, iron ore and other minerals. The belt conveyor system may include the following version in surface mining system-

Face →Hopper → Crusher → Belt conveyor → Dump yard or material handling plant

Face →Hopper →Crusher →High angle conveyor → Belt conveyor →Dump yard or material handling plant

Crushing Plant →Belt conveyor →Material handling plant →Consumer plant or industry, etc.

The longest length of conveyor presently available in the world is around 100 km.

The basic parameters of selection of belt conveyor system is as follows:

- Amount of material (t/h) to be handled both in average and in peak load condition,
- Basic characteristics of the materials like density (to calculate volume or tonnage), maximum lump size (it should be less than the width of the conveyor), amount of fines, watery condition, stickiness, roughness, abrasivity, temperature, dust condition, chemical characteristics, etc.
- Surface topography and site condition (locality, township, etc.)
- Availability of electricity in the site, type (whether A.C. or D.C.), frequency, voltage and current condition, availability of two sources of electric power in the site,
- Climatic condition and their adaptability in that condition,
- Working time of a conveyor,
- Cost economy.

Advantages

Belt conveyor system has the following advantages-

- Highly reliable,
- Specific power consumption is very less,
- High capacity,
- Negotiate a considerable gradient (ordinary belt conveyor 18° and high angle conveyor more than 60°).

A belt conveyor may be used at a gradient of up to 18° for lumpy ore but for finely crushed and powdery material the angle of inclination may go up to 27°. It is always advisable to use the belt conveyor from flat condition to around 10° to 12° for obtaining good result. Presently high angle belt conveyor has been introduced to negotiate an angle up to 60°. The angle of slope is selected lower than the angle of repose of the materials. It depends upon the belt speed, idler spacing, riding ability of the materials, etc. which again depend upon the size, shape and condition of the material.

Down Hill Conveyor Run

Maximum belt conveyors are selected to convey the material in the uphill side; although there are many instances when the belt conveyor transports material in the downhill direction particularly in the hilly terrain. In that case except during starting operation to overcome the frictional resistance between the belt and the idler either very less power or no power is necessary (depending upon the degree of belt inclination) to transport materials by the belt conveyor. While belt running with no power special thruster operated braking arrangement with optimum force is to be provided in the belt conveyor system to check the momentum gained by the material along with the belt for proper control and stoppage. The momentum of the belt depends upon the speed of the conveyor, size, type and weight of the material, inclination, watery condition, and other characteristics of the material. During selection of brakes the heat generated by the frictional resistance is also to be taken into account. The installation cost in this system is very less and the power generated by this system can efficiently be utilized for other purposes provided a motor of sufficient size is used. Feeding onto the belt conveyor should be properly controlled by a discharge feeder conveyor. If the load exceeds beyond the automatically controlled matching the designed capacity of the downhill conveyor. The feeding rate should be designed capacity of the conveyor , the braking system will be overloaded and belt will run way. The size of the material is very important. It assists to select the belt width, speed and the angle of inclination of the transportation system. Clay material if present and if it mix with water make the ore or mineral very sticky which also affect the amount of material transport. If the downhill belt conveyor stops suddenly , there may be the chances of huge amount of spillage of materials from the conveyor.

DESIGN OF BELT CONVEYOR

Design of a belt conveyor includes the following parameters-

- Width and speed of the belt,
- Motor power,
- Idler size, shape, design, spacing, etc.,
- Drive type and layout,
- Construction of belt,
- Size of belt drums and pulleys,
- Belt conveyor structure and other accessories,
- Belt take up system.

Width of the Belt

It is selected based on the following considerations-

- Required capacity,
- Maximum lump size,
- Cost (cost of the belt accounts about 50% of the total cost of the belt),
- Riding ability,
- Material weight,
- To prevent side spillage it must be of sufficient in size,
- Speed of the belt (speed should be as fast as possible. If the speed is high, the width of the belt may be kept narrower).

The minimum belt width for 20° troughing angle according to the German standard is given as follows-

Maximum lump size (mm)	100	150	200	300	400	500
Minimum belt width (mm)	0.4	0.5	0.65	0.8	1.0	1.2

The American conveyor equipment manufacturer's associations recommend the following-

For a 20° angle of surcharge with 10% lumps and 90% fine, maximum lump size is - $\frac{1}{3}$ rd the belt width.

For a 20° angle of surcharge with all lumps and no fines maximum lump size is $\frac{1}{5}$ th the belt width.

For a 30° angle of surcharge with 10% lump and 90% fine maximum lump size is $\frac{1}{6}$ th the belt width.

For a 30° angle of surcharge with all lumps and no fines maximum lump size is $\frac{1}{10}$ th the belt width.

In Russia the following empirical relationships are used-

$B \geq 2\, a_{max} + 200$ (mm), for non sized material

$B \geq 3.3\, a_{average} + 200$ (mm), for sized material

The standard belt widths manufactured in the world in millimeter are-

400,500,650,800,1000,1200,1400,1600,1800,2000,2200, 2400, 2600, 2800, 3000, 3200 .

The economical belt width range varies from 900 mm to 2000 mm.

Belt Speed

The speed of a belt depends upon the following-

- Peak load demand,
- Weight of the material,
- Belt width,
- Length of belt,
- Idler spacing,
- Motor power,
- Strength of the belt,
- Nature of the material,
- Maximum lump size.

The speed range (m/s)

For picking belts	0.1- 0.3
Stationary conveyors (underground, limestone and cement plants on the surface)	1.8 - 3.0
Heavy duty long conveyors	4.0 - 8 .0
Belt feeders for feeding mildly abrasive materials from hopper and bins	0.25- 0.50
Belt with trippers driven by the conveyor belt itself	2.0

However the belt speed 2 - 3 m/s at a belt width 600mm to 1000 mm is considered normal.

Motor Power

The requirement of motor power is to overcoming the frictional resistance to the motion of the conveyor parts and any accessories operated by it. It includes over coming of

- the frictional resistance to rotate the idlers,
- the frictional resistance of the belt due to flexure as it moves over the idlers,
- the frictional resistance due to impression of the belt on idlers,
- the sliding resistance between the belt and idler rolls due to misalignment,
- the frictional resistance of the load to flexure (shifting and reshaping of load) as the belt and load move over the idlers,
- the frictional resistance of the terminal and bend pulleys (drums) to rotation (frictional resistance of the bearings of the pulleys),
- the frictional resistance due to belt wipers on pulleys
- the frictional and inertia resistance at the loading point(s),
- the frictional resistance due to conveyor accessories such as tripper and skirt boards, etc.

Motors may be either squirrel cage induction motor or slip-ring induction motor and should not deliver power 50% in excess of its continuous rating during starting and acceleration period. This characteristics are achieved either by increasing the acceleration period in the long conveyor by using time-control multi-step resistance starters in case of slip ring multi-motor drives. For a 4-motor drive 20 s is necessary as a starting time. But with squirrel cage induction motors (having short starting time characteristics), a mechanical or hydraulic coupling or torque converter should be used in between the motor and the gearbox system. The squirrel cage induction motors are easy to maintain, cheap and easily available. In multi-drive system an in bye pulley motor should be started followed by the out bye pulley motor after a delay sequence of 2 - 5 s, otherwise slack belt will be thrown off between driving pulleys causing the pulley to lose and gain the traction alternately inducing belt surge.

Fluid Coupling

Fluid coupling (Fig - 7.5) may be divided into two types viz (a) Traction type fluid coupling (b) Scoop control type fluid coupling. The traction type fluid coupling consists of a fluid chamber to hold either oil or water. The fluid chamber consists of an impeller unit driven by a motor and a runner unit connected to the gearbox unit. When motor starts rotating, it rotates the impeller unit. Because of this rotation, the fluid inside the coupling will also rotate by the centrifugal action created by the vanes mounted with the impeller. The fluid then impinge the stationary runner unit with a high speed and afterward flow back again to the running impeller to create a vortex. As the process continues sufficient amount of kinetic energy is developed by the impeller transmits into the fluid. Higher the speed of the impeller higher would be the development of kinetic energy in the fluid. Ultimately the kinetic energy will increase very high and will run the runner unit. Both the runner and impeller units contain vanes set radially and are housed by a casing to hold the fluid inside them. A protective fusible plug containing alloy plunger is connected with the fluid coupling, which melts when the temperature in the fluid exceeds it's melting temperature (100°C to 140°C). When the alloy plunger melts, the fluid inside the coupling ejects out leaving the fluid rotating without load. Thus the motor and other sensitive mechanical units are saved from burning out. Beside the fusible plug ,a bursting disc is connected with the fluid coupling which has the same purpose like a fusible plug but acts when the later fails to operate. The coupling is designed to have a torque transmitting capacity around 2.2 times the full load torque. Normally the transmitted torque is obtained with a slip of 2% to 3%. The basic purposs to use the fluid coupling are as follows :

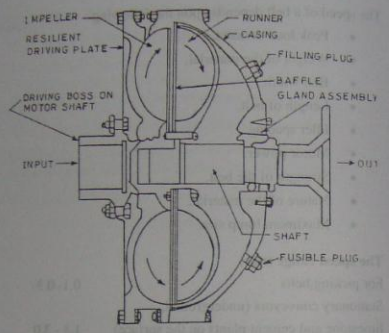


Fig. 7.5 : A Schemetic Line Diagram of a Fluid Coupling

- To reduce shocks stresses and loads on the motor and power transmission unit particularly during starting which reduce much of electrical and mechanical failure,
- To improve the starting capability and starting characteristics,
- To distribute the loads uniformly among the motors in the multi-motor system,
- To operate the machine safely under full load, severe vibration and impact conditions, etc.
- To simplifies the design of much complicated electrical equipment,
- To separate out easily the motor and gearbox units .

In the scoop control type fluid coupling a reservoir casing surrounds the casing where oil is contained below the level of opening. Beside runner and impeller unit a sliding scoop tube is carried within the rotating reservoir casing. The open mouth of the scoop tube faces towards the rotation of motor. The scoop tube controls and completes the oil circuit. If the fluid coupling is allowed to be driven by the motor and the scoop tube is withdrawn either by a manual, automatic or remotely controlled operating lever, oil will only form an annular rim in the reservoir casing. The oil circuit in this case will be incomplete and empty, resulting no power transmission in the gearbox unit. But when the scoop will remain in it's proper position , the working oil circuit will complete and the power will transmit in the gearbox unit. By suitable adjustment of the scoop lever the speed can be varied and controlled and hence the torque is also controlled accordingly.

This type of coupling has the following extra merits as compared to a traction type coupling-

- By changing (by vertical sliding) the position of scoop tube the torque can be varied in wide range,
- The acceleration of speed is very smooth,
- Wide range speed regulation is possible,
- The motor can be disconnected easily from the machine while the former is under running condition,
- Automatic control is possible.

Belt

A belt mainly consists of two parts viz (a) Carcass and (b) Cover.

Carcass

It is the backbone of the belting consisting of several specially woven textile plies. These plies are bonded together either by natural rubber or by synthetic rubber. The warp and weft can be of the same yarn or different yarn or combination of different yarns like cotton, nylon, polyester, cotton/nylon, rayon/cotton, rayon/nylon, etc. Carcass constructions are specified by the number of plies, the type of fabric actually used in each ply in the belt etc. A 5 ply, fabric CN 200 represents a carcass having 5 plies each employing cotton/nylon in fabric and the tensile strength of the fabric in the longitudinal direction is 200 kN/m/ply. Between the plies an extra thickness of rubber coating may be added up for improving inter-ply additions and for cushioning to the impact and heat. Nylon fabric costlier as compared to a cotton fabric but offers the following main advantages-

- More flexibility,
- High resistance to longitudinal tearing resistance due to impact affect,
- Lighter in weight,
- Have good fastener holding properties,
- Efficiently be used on deep troughing idlers.

Cotton fabrics are more susceptible to moisture and more prone to rotting on account of fungus formation. In cotton/nylon belting, the cotton bulk up the body and the nylon provides the strength. It is cheaper than nylon fabric belting plus it has all the advantages what a nylon belting posses.

Ply constructions are as following-

Stepped ply-In this case the flexibility of the belt in the transverse direction is more for better troughing and the cover thickness in the centre is more to negotiate more wear.

Breaker on edge-The breaker ply in this case is folded around the edges to give higher protection of edges of the belt. (Fig - 7.6)

Straight ply-Plies are arranged in the straight fashion. (Fig - 7.7)

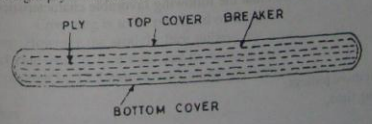


Fig. 7.6 : Breaker on Edge Construction Belting

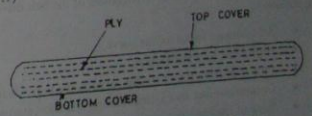


Fig. 7.7 : Straight Ply Construction Belting

Breaker ply-An extra specially woven ply is placed in the cover to provide belt a greater resistance to impact due to falling pieces. (Fig - 7.8)

Steel cord belting (Fig - 7.9)-In this case special cables or cords of nylon, cotton, rayon or steel are placed in the belt to take the longitudinal load. Duck plies are also embedded into it for withstanding transverse strength. However they are not considered as load carrying plies. The diameters of the steel cords are around 1.2mm to 9.5 mm.

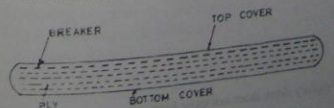


Fig. 7.8 : Breaker Construction Belting

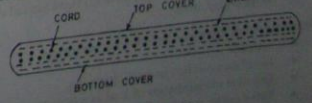


Fig. 7.9 : Cord Belting

The steel cord belting has the following merits :

- It has better troughing characteristics,
- Idler spacing are wider, hence structural cost is less,
- Longer life,
- Smaller pulley (drum) diameter (around 25 times the cord diameter),
- Excellent impact, moisture and heat resistant,
- Lower stretch factor of 0.2% at maximum working tension. Hence mechanical winches can be used for the take up system.
- Possibilities of higher speed,
- Take-up distance is shorter,
- It can nicely be used for an average distance around 5000 m, however even the conveying distance in one flight may be around 15 km with a belt speed of around 8.4 m/s.

Cover

Rubber, P.V.C. or plastic covering does not provide any strength but protect the fabric from abrasion, wear and action due to moisture or water. In P.V.C. resistance due to weir and abrasion is higher as compared to the rubber covering but it's resistance to cutting or gouging resulted due to impact force by large and heavy material are relatively less.

The construction of the belt carcass mainly depends upon the requirement of tensile strength and this tensile strength depends upon the tight side tension. Accordingly the number, type of ply, etc., are selected. But the following parameters are also very important to select a belt-

- Transverse and longitudinal flexibility,
- Resistance to moisture,
- Resistance to ripping, tearing, impact, etc.

Types and General Arrangement of Drives

The types and general arrangement of the belt drives would be such that the following favorable characteristics are obtained-

- An optimum speed torque curve,
- High reliability and safety,
- Proper distribution of torque between the individual drive pulleys,
- Continuous and shock-free running and short starting time,
- Ease of maintenance by unit replacement,
- Easy to combat overloading of drive motors.

A general arrangement of belt conveyor drive has been shown in the Fig - 7.10.

Types of Drives

- Single pulley drive
- Two pulley drive-It is divided into the following-
  (1) Tandem drive (2) Duel drive (3) Head and tail drive.
- Three pulley drive.

The single pulley drive has the following advantages-

- Simple,
- Less flexing of belt under high tension,
- Question of distribution of load as arise in the multi-pulley drive does not exist here,
- Drive pulley is in the clean side of the belt,
- Easy to maintain and repair.

The multi-pulley drive has the following merits-

- Requirement of belt tension is less,
- Size of pulleys are smaller,
- Belt thickness is less,
- Size of the gearbox is less,
- Use for longer distance haul which require higher power.

The most commonly used drives are dual - pulley drive. The advantages of duel pulley drives are as following :

- Drives are very compact and interchangeable,
- Drive pulleys are in the clean side of the belt, easy to maintain and repair,
- A belt conveyor can be converted to a higher duty by adding extra motor,
- Whenever a drive becomes out of order, the belt may run with reduced capacity by only one drive.

The head and tail drive pulleys has the following advantages-

- It can be used under large variations in loading condition,
- It can be used over the undulating profile,
- The tail drum produces a certain amount of tension and reduces the tensile stresses to which the carrying belt is subjected during starting,
- Reduces the amount of take-up weight,
- Reduces the belt conveyor cost.

Belt Pulleys (Drums)

The diameter of the belt pulleys (drum) depends upon the thickness of the belt, number of plies, tension, length, life of the belt, etc. Larger diameter belt pulleys (drums) are selected for longer length and higher life. The diameter of the pulley (drum) is around 100 times and 150 times the thickness of the belt and ply respectively. However the minimum diameter of pulleys in high tension and low tension are 400mm to 600 mm and 250mm to 350 mm respectively.

The recommended widths of the pulleys are as following-

For belt width up to 1070 mm	Belt width + 50 mm
For belt width over 1070 mm	Belt width + 75 mm

Crowning is to be practiced on all drive pulleys where the tension in the belt is less than 40% of maximum belt rating. However the straight face pulleys are mostly used in dual or tandem drives at any belt tension level. To reduce slack side tension, face wear and to increase grip between the belt and the pulley and self-cleaning action, the belt pulleys are covered by 12.5 mm vulcanized herringbone or chevron-grooved or plain cover lagging.

Idlers

The belt is supported by rollers or idlers for it's smooth movement by reducing excessive frictional resistance. The top idler sets (2, 3 or 5 in numbers) consists of a central idler and troughing or side idlers to provide sufficient angle of trough to the belt. Most of the top idler set consists of 3 number of idlers per set (Fig - 7.11(A) and 7.12). Bottom idlers are mostly straight which is a little bit longer than the belt width.

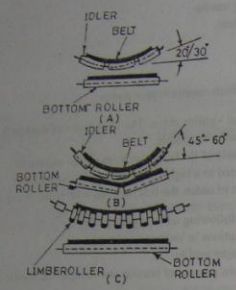


Fig. 7.11 : Shape of the Belts

The idler roll is designed based on the following characteristics of the material,

- Belt speed,
- Operating condition,
- Life of the conveyor, etc.

Balance, sealing, shaft deflection, load bearing capacity, etc. are some of the main design characteristics of the idlers. A good idler design reduces wear on belt and idlers and lower down the requirement of power. Idlers are fitted with either ball or tapered roller bearings and non rubbing labyrinth type metal seals on either side of each bearing. All idler rolls are grease filled for life. The outside diameter of the idlers are 65mm, 90mm, 108mm, 133mm, 159mm, 191mm etc. To transport highly abrasive materials 159mm idlers made of cast iron or rubber covering over the steel rolls are used to increase the life of the later. The rubber covering also provide cushioning effect on the rolls against the impact forces created by the falling of big lumps. Sometimes the idlers worn out with rubber coverings are termed as impact idlers. In most of the belt conveyors, the impact idlers are provided in the loading points. To combat attack by the acids under wet operating conditions ultra high molecular weight polyethylene sleeves are used instead of rubber covering. The spacing of the carrying idler sets depends on the weight of the belt and material and amount of sag of the belt in between the idlers. If the spacing between the idler sets are more ,there will be excessive sagging of the belt causing sharp creasing of the belt at idlers which lead to excessive flexure of the belt, requires more motor power, causes spillage over the belt edges and overloading of idler bearings and higher impact forces on the idler by the big boulders.

The spacing between the carrying idler set varies from 0.9m to 2 m depending on the density of the ore or rock. Recommended spacing : 1m for iron ore , 1.2m to 1.3m for crushed limestone , stone and coal. The return idler spacing varies from 2m to 4m. The most common range of spacing is around 3m with all belt width.

In some of the belt conveyor self alignment idler sets (Fig - 7.13) are installed at considerable intervals to prevent swaying of belt by swiveling the self alignment belt structure with the help of a ball bearing. The structure contains three top idler along with vertical side idler at each side.

Belt Shape

The standard troughing angle for 3-roll idlers is 20° and 30° (Fig - 7.11(A)). The 30° troughing angle is most commonly used. Overloaded conveyors have troughing angles of 30° or 35°. For low height, flat belt, horizontal idler are used for a shorter length belt conveyor with a large angle of repose. This flat belt conveyor has high specific power consumption and suffers from larger wear. To carry bulk light weight materials on narrow belts 2-roll troughing idlers may be used. The loaded cross sectional area in this case is more as compared to a 3 roll idlers having 20° troughing angle and save much of cost.

The garland type (resiliently suspended type) idlers offer variable troughing angle (Fig - 7.13). Idler rolls in this case flexibly coupled together by chain links, or limberoller (Fig - 7.11(c)) which consists of a series of neoprene discs molded to neoprene-sheathed flexible steel cables each of which is freely suspended from two ball bearings mounted on stands or brackets so that the belt shapes itself to a load-conforming catenaries.

The advantage of the variable troughing are as followings-

- The belt can be trained under any load condition,
- Higher carrying capacity,
- Higher life of the belt and idlers,
- Building-up of material is reduced,
- Less maintenance requirement,
- Limberollers can resist most chemical action and also can handle abrasive materials.

Presently the deep troughing (3-roll deep-trough idlers) with side roll inclinations of 35°, 40° or 45° and 5-roll garland type idlers (Fig - 7.11(B)) with 30°/45° troughing are most commonly in use.

The advantages of deep troughing are as following-

- Carrying capacity will increase,
- Reduce spillage,
- Reduce the width of a belt for a given tonnage,
- Less wear on the cover per Tonne of material handled,
- Requirement of skirt board is totally eliminated,
- Spacing of idler would be wider,
- Capital cost will be reduced.

The following points are to be considered in depth during a belt installation-

- Proper care is to be taken so that the belt edges do not severely overstressed. The transition distance from the last troughing idler to the terminal belt drum should be as following-
  For 35° idler set : 1.5 x belt width
  45° idler set :2.0 x belt width
- To prevent overstressing or wrinkling of the belt edges, the radius of concave or convex curve should be large enough. The pitch of the idlers should be reduced to less than half the normal pitch at the centre of the curve.
- Transfer or loading points must be thoroughly designed to ensure full loading of the conveyor.
- It is always better if the speed of the belt is sufficiently high,
- The carcass must be flexible enough to conform to the troughing. Carcass of solidly-woven construction is suitable for deep troughing.
- High stresses can be set up in the belt at the point of flexing between the centre and side rolls and also by providing correct gap between the centre and side rolls,
- Use synthetic fabric or steel cord belts for deep trough idlers,

Sometimes a belt conveyor may be served as bunker by using 5-roll idlers with 30°/60° troughing.

Belt Cleaning Devices

The basic purposes of belt cleaning devices are as following -

- To remove caking and incrustation on the carrying belt surface,
- To remove any material that may spill onto the upper side of the return run of belt,
- To increases the belt life
- To eliminate materials build-up on return idler rolls and pulleys otherwise it may cause belt misalignment
- To reduce conveyor downtime
- To reduce the maintenance cost.

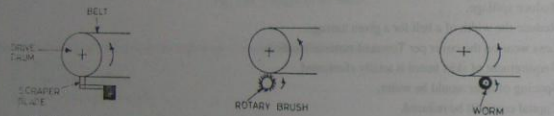
There are many varieties of belt cleaner out of which the followings are worth to mention-

- Single or multi-wiper blade scraper
- Rotary brush
- Water spray and wiper blade

Single or Multi-Wiper Blade Scraper [Fig - 7.14(A)]: It consists of one or more rubber blades clamped between two steel strips and make contact with the belt surface by counterweight or spring tension and scrapes material from the belt surface. A good quality scraper blade is made from a special quality rubber made from three layers (Combi scraper blade) of different shore hardness having a central soft layer embedded between two outer harder layers.

Rotary Brush [Fig - 7.14(B)]: It is a power driven tube to which nylon or other soft material brushes (bristles) are attached. Generally they are driven by chain from the adjacent head pulley shaft or by an independent motor of 0.55kW to 1.5 kW. They are designed as low speed (peripheral speeds of 2 m/s to 3 m/s) or high-speed (having peripheral speeds of 5m/s to 7.5 m/s) type depending upon the nature of the incrustation. The rotational direction of the brush is in the direction opposite to the running direction of the belt. The high speed of rotation is around 900 rpm.

Water Spray and Wiper Blade : Sometimes in difficult condition high pressure water is used as an effective mean of belt cleaning. Beside above, 'Worm Type' belt cleaner shown in the [Fig. - 7.14(C)] is also used for cleaning the belt.



Blade Type Cleaner (A) — Brush Type Cleaner (B) — Worm Type Cleaner (C)

Fig. 7.14 : Schematic Line Diagrams of Various Type Cleaners

(C) Bottom Cleaner (Fig - 7.15)-Sometime materials accumulates over the bottom belt by spilling from the top belt and it carries up to the tail drum when it causes belt misalignment, damage of belt and pulley. Hence a plough type bottom cleaner made of steel channel is provided to clean the material from the belt.

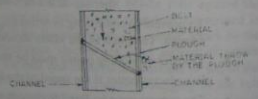


Fig. 7.15 : Bottom Cleaner

Belt Take-Up System

The purposes of the belt take up system are as following-

- To stretch and shrinkage of the belt.
- To take up the slack belt which would otherwise cause the belt to sag.
- To store sufficient quantity of extra belt (around 100m to 120 m) for extension or reduction of conveyor length.
- To maintain adequate tension in return belt during starting up and under varying load conditions to ensure sufficient frictional grip between the drive drum and the belt.

Belt stretching will vary with temperature, atmospheric conditions and tension.

Types of Take-Up System

(1) Manual Take-Up
   (a) Screw take-up (b) Winch type loop take-up (Fig - 7.16) (c) Jack take-up.
(2) Automatic Take-Up
   (a) Gravity take-up (b) Automatic motorized winch and load cell loop take-up (c) Pneumatically or hydraulically operated loop take up.

Manual screw take up and jack take up is generally used for light duty short centre distance belt conveyor around 300 m length. In screw type take up system the coupling will be very long for longer length belt conveyor and the long screw may get bend. Gravity take up is generally of two types viz. vertical take-up (Fig - 7.17) and horizontal take-up (Fig - 7.18). The gravity take up system is generally installed when sufficient rooms are available. It is very simple.

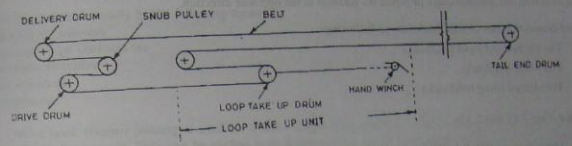


Fig. 7.16 : A Schematic Line Diagram of Manual Loop Take up Unit

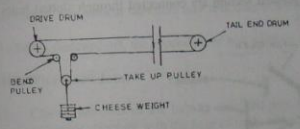


Fig. 7.17 : Gravity Take up (most commonly used)

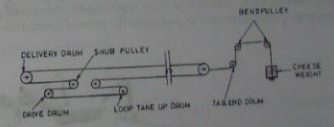


Fig. 7.18 : Another Version of Gravity Take up

economical and have constant force and also the speed in it is instantaneous. Take up pulley generally located at the tail end of the ascending conveyor (some are located near to the drive head) which is not fixed and it moves downward due to counter weight giving tension to the slack belt. But the height of the tower requirement will be very high in this case.

The horizontal take-up system requires more horizontal room and located at the tail end side of the belt conveyor. It is not economical due to containing more accessories.

The power operated automatic take ups are most desirable on high speed long conveyors which produces maximum amount of slack belt during starting up. An automatic electric take-up basically consists of a load-cell and a motorized roped winch (Fig - 7.19). The load cell is designed to suit a predetermined belt tension for normal running condition. At a certain percentage above or below this tension, the load cell sends a signal to the motorized winch either for pulling in or paying out of the rope from the winch connected with the load cell to the take-up carriage. About 100 - 120 m belt storage is possible in the modern belt conveyor system.

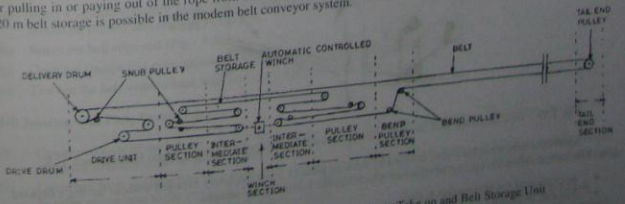


Fig. 7.19 : A Schematic Line Diagram of Automatic Loop Take up and Belt Storage Unit

Holdbacks

Holdbacks are connected with the drive shaft of the uphill moving belt conveyor system when the inclination of the conveyor is more than 6°. It is a holding device to stop the run back of the conveyor in case of stoppage of the power while the conveyor was on forward travel with load. A holdback allows free rotation of the drive pulley shaft in the forward direction but automatically prevents it's rotation in the opposite direction.

The most common types of hold backs are as following-

- The ratchet and pawl holdbacks
- The roller holdbacks
- The shaped sprag holdbacks

Structure (Fig -7.12 and 7.13)

The conveyor structure is of stringer type as the part can be easily erected and dismantled. The troughing idlers and return idlers are supported by this structure running throughout its length. Commonly used structures are made of mild steel channels, although in some types angles and special pressed section are connected through slotted holes to negotiate a slight difference in gradient at the point of connections.

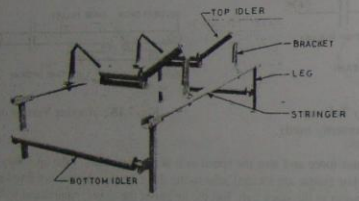


Fig. 7.12 : Belt Structure with Ordinary Idlers by Courtesy Meco International, U.K.

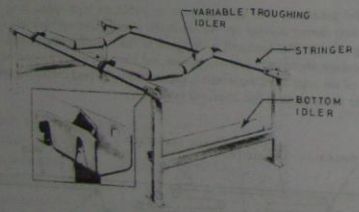


Fig. 7.13 : Belt Structure with Modern Variable Troughing Idlers by Courtesy Meco International, U.K.

Conveyor structures are either covered by decking plate or it is completely exposed without having a decking plate. The later is called the open-type structure which has the following advantages-

- Inspection of the bottom belt is easy
- The risk of heating due to the accumulated coal is less.
- Cheap
- Easy to erect and handle

The decking plate collects the side spillage falling off the top belt onto the bottom belt and hence materials are prevented to flow in the tail end drum and thus save belt conveyor from the breakdown.

Belt Jointing

Belt jointing is done basically by the following means-
(1) Mechanical Fasteners
Most common type fasteners are
(a) Wire hook fasteners
(b) Plate fasteners
(2) Vulcanizing

Wire hook fastener jointing is used for smaller length belt conveyor whereas the plate fasteners are used for longer length belt conveyor. They can be applied both in the surface and in any underground mining system. However the mechanical fasteners joint has lesser strength as compared to the original belt and it is the weakest point of the belt conveyor system. Belts are more frequently snatched off in this joint. Some amount of fines also leak through the joint. Jointing by mechanical fasteners take less time as compared to the vulcanizing system.

If a belt is snatched off, the procedure of belt jointing will be as follows-

With Wire Hook Joint

(i) Clamp two ends of the belt by separate steel clamps leaving 2 m from the end points. Tie up one clamp to the conveyor structure with the help of chain and bolts.

(ii) Loose belt from the loop take up system. Pull one end manually with the help of a tirfor. The other end of the belt to be gripped by another clamp near to the end of the belt gripped by the first clamp. The either end of the tirfor rope is attached with each of the clamp with the help of chain and bolts.

(iii) Lay one end of the belt on top of the other and mark the points with chisel where the belt ends are to be cut.

(iv) With the help of a carpenter type square draw a line perpendicular to the centre line at the points marked in both ends of the belts. Cut the belt ends at these lines with the help of a chisel and a hammer.

(v) Insert paper combs (in which the wire hooks are mounted) in the jaws of a belt fastening machine which presses all the hooks in the belt end with equal pressure, at the same extent and same spacing perpendicular to the belt axis leaving around 50 mm from the sides of the belt. Insert belt end fully into the open jaws of the hooks, operate the machine to close the jaw and press hooks into the belt making a wire loop at the end edge of the belt.

(vi) Do the same procedure for the other end.

(vii) Belt ends thereafter are pulled together so that hook loops are intermeshed.

(viii) Insert a spindle specially made from flexible stranded rope into the intermeshed hinge. The length of the spindle should be slightly less than the belt width.

(ix) Notch the belt edge end (Fig - 7.20)

(x) Open the clamps and give optimum tension to the belt by the loop take up system.

(xi) Test run the belt with no load, half load and finally with full load.

Belt Jointing with Plate Fastener Joints

(i) Clamp two ends of the belt by separate steel clamps leaving 2 m from the end points. Tie up one clamp to the conveyor structure with the help of chain and bolts.

(ii) Loose belt from the loop take up system. Pull manually with the help of a tirfor the other end of the belt gripped by other clamp near to the end of the belt gripped by the first clamp. The either end of the tirfor rope is attached with each of the clamp with the help of chain and bolts.

(iii) Lay one end of the belt on top of the other and mark the points with chisel where the belt ends are to be cut.

(iv) With the help of a carpenter type square draw a line perpendicular to the centre line at the points marked in both ends of the belts. Cut the belt ends at these lines with the help of a chisel and a hammer.

(v) Butt the belt ends with each other and nails a template supplied by the manufacturer of the plate fastener accurately in position.

(vi) Take a bore-hole punch matching the template holes and punch boreholes in both belt ends in accordance with the holes designed in the template.

(vii) Withdraw the template.

(viii) The bottom plates having bolts are assembled in the bottom of the butted portion and one number of bolt of each plate is inserted in each belt ends.

(ix) Place top plates over the bolts of each bottom plate and tighten the plates fully with the help of nuts. If requires hammer the plates.

(x) Cut off extra length of the bolts over the nuts.

(xi) Hammer the bolts again.

(xii) Notch the belt edge end (Fig - 7.21).

(xiii) Open the clamps and give optimum tension to the belt by the loop take up system.

(xiv) Test run the belt with no load, half load and finally with full load.

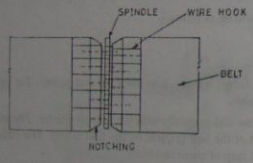


Fig. 7.20 : Hook Jointing

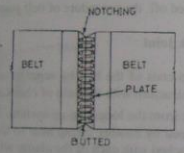


Fig. 7.21 : Plate Jointing

Vulcanized Jointing

Vulcanization is a process of jointing two ends of the belt by synthetic glue to make it endless. It has the following advantages-

- The joint does not give rise to any leakage of material through it.
- Belt joint is very smooth, strong, has higher life and also the joints has no impact affects with idlers drums or belt cleaners.
- Belt tearing from the joints is not possible.
- Cheaper belt with less number of plies can be well employed.

But since vulcanizing machine is neither flame proof nor the intrinsically safe, it is not permitted in the gassy degree -III coal mines in India.

Splicing of belts can be done in many ways which are as following-

(a) Diagonal Stepped Splice: The successive plies at the end of the belt are cut in this case along a line of around 26° through the axis of the belt in the stepped manner. The stepping in the other end of the belt is reversed so that when both ends are pasted with synthetic rubber cement solution, embedded with each other and vulcanized, they act as a continuous ply and become an endless one.

(b) 'V' Splice : In this case the cut is made in the form of a 'V' and the successive plies are also cut in 'V' shape. The stepping in the other end of the belt is also cut in 'V' shape but in reverse order so that when both ends, after pasting with synthetic rubber solution, embedded with each other and vulcanized , they act as a continuous ply. Making of this type of plying is however very difficult, time consuming and needs greater degree of expertise for vulcanization. Belts in such case train effectively.

Vulcanizing Procedure

The procedure of vulcanization is as follows:

(a) With the help of a carpenter square draw the edge line perpendicular to the centre line in both the ends of the belt. Cut off the ends at these lines.

(b) Make either diagonal stepped splice or 'V' splice cuttings in both the ends of the belt after cutting off the each ply in accordance with the table supplied by the belt manufacturer. The belt manufacturer mention the width of the steps in the plies, total number of steps, the length of the splice depending on the size of the belt, number of plies, type of fabric of the belt, etc., in the manufacturer catalogue.

(c) Remove all the rubber of the top and bottom cover of the belt with the help of a hand grinder.

(d) Apply synthetic cushion gum over all the exposed cut surfaces with the help of a roller followed by the brushing with rubber cement solution.

(e) Match accurately by placing the two ends over one another and apply cover compound over the exposed cover portion of the top and bottom of the belt surface. The edges are also covered with the cover compound strips.

(f) The jointed belt portion thereafter is spliced in between the heating/pressing plates of the vulcanizing machine. The heating/pressing plates are correctly positioned and aligned by the guide supports on the heating/cooling element. The bottom heating plate is placed over the bottom modules first, thereafter the belt splicing section and afterward the top heating/pressing plate is put over the belt section. Over the top plate, top pressure traverses are placed and are tightened to a predetermined pressure (6.8 - 17 kg/cm^2).After connecting all plugs to the terminal strip on the electric supply line, heat the heating plate to a predetermined temperature recommended by the manufacturer for a particular duration varying from 30 - 50 minutes.

(g) Switch off the electric current after completion of the vulcanizing period. Supply cooling water through the hoses connected to the channels of the heating/cooling elements. The water evaporates and results in a very effective cooling. The cooling process is finished when no more steam escapes which means that the temperature of the press having cooled to below 100°C.

(h) After dismantling the vulcanizer allow to cool the belt further for a few hours before use.

Examples

1. KL IV Module Vulcanizer Manufactured by Wagener Schwelm : It is used for splicing and repairing of conveyor belts width up to 1400 mm. It has fully automatic ceramic heating system with hydro-mechanical pressure system for uniform pressure.
2. KL IV Combi Vulcanizer Manufactured by Wagener Schwelm : It is used for splicing and repairing of conveyor belts width from 1400 mm to 3200 mm.

BOOSTER BELT CONVEYOR

To increase the length and capacity of a belt conveyor, one or number of secondary belt conveyors are inserted inside the main belt conveyor. It is done in such a manner that the secondary belt conveyors' endless belts come in direct contact with the underside of the main belt which increases power at the intermediate points along the length of the main belt conveyor. It helps distribution of power evenly whole along the main belt conveyor by frictional contact with the main belt and because of this reason low strength belting can most efficiently be used in the main carrier belt. Loads are shared by the intermediate drives and this arrangement lower down the belt tension as compared to an ordinary belt conveyor (Fig - 7.22).

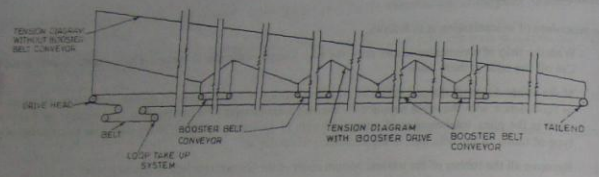


Fig. 7.22 : Schematic Line Diagram of Tension Curve with and without Booster Belt Conveyor.

Advantages

Booster belt conveyor system provides the following main advantages-

- Conveyor length and capacity can be increased.
- Since belt tension is reduced, lower tensile strength belt can be used.
- Requirement of power will be less.
- Since length of belt conveyor (single flight) is increased, number of transfer points will be less.
- Increase the flexibility of operation in the belt conveyor system.
- The belt conveyor system will keep running in case any booster belt conveyor remains under breakdown.
- The physical size of the motor, gearbox, drive head and other main transmission accessories are less bulky and if standardization of all the drive units are done , it will reduce number of spare parts items and hence it will solve the various problems of the material management department.
- Capital cost will be lowered down due to the savings on belting cost and elimination of other costly accessories which would have been otherwise needed if ordinary belt conveyor has been used.
- It minimizes excavation cost to accommodate the drive unit.

Some Salient Points

In booster belt conveyor system, the main belt unit is started first. Then intermediate booster belt conveyors are started one by one after a short interval of time depending upon the distance and power. It is always better if the drive units of the main belt conveyor and the booster belt conveyors are standardized so that they have equal load sharing characteristics. Special control and monitoring equipment are necessary for smooth starting of main belt and booster drives and also for proper control and monitoring of the total system. The width of the booster belting shall be at least 15 cm less as compared to the main belting and the length of the booster belt conveyor depends on the co-efficient of friction between the two belts, material to be hauled, gradient of the surface profile, etc. Number of intermediate booster belt conveyors may be one to more than four and the length of main belt conveyor (in single flight) incorporating the booster belt may be more than 6 km; having transporting capacity varying from 500t/h to more than 1600 t/h; depending upon the width and speed of the belt, gradient of haul, power of the drive units, etc.

CABLE BELT CONVEYOR

The belt in this case does not attach to the driving medium but it is merely rest and driven by two endless cables (ropes). The specially constructed belt is molded with shoes which rest over the cables, serves only the purpose of supporting the materials to be conveyed. The cables transmit the driving pull. No troughing idlers are provided to support the belt but cables are supported by grooved pulleys mounted on line stands set at regular intervals throughout the length of the conveyor (Fig - 7.23). At the delivery and tail ends, the belt is separated from the wire ropes with the help of specially adjusted skew pulleys which offset the cables and belt supporting idlers (having lining) raise the belt from ropes before reattaching with them again. Both the cables are operated by the same drive unit. The cable is made of either steel wire rope of langslay construction with a fiber core, having diameter ranging from 22 mm to 62.5 mm or

cable have a wire stranded core. Both the core and the cable may be covered with a synthetic rubber to increase the flexibility. The ropes containing galvanized steel wires are manufactured in longer length (more than 10 km) to minimize the number of splicing (the splicing length is generally 1000 times the diameter of the rope). Average life of a rope is generally 2 to 3 years, however it can last for more than 5 years depending upon the operating conditions and hours of operation.

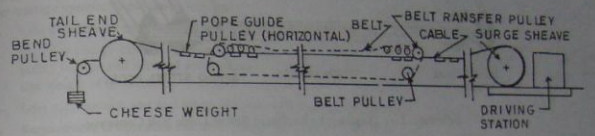


Fig. 7.23 : A Schematic Layout of a Cable Belt Conveyor.

Endless cables (ropes) run over the supporting pulley of diameter varying from 300mm to 400 mm. The pulleys are attached with the line stands which are pitched at a distance varying from 2m to 15 m (average 7m to 8 m) depending upon the contour of the belt conveyor (Fig - 7.24). Line stands are made by mild steel frame and fixed over either on pre-cast concrete sleepers placed on the ground surface or may be bolted to the floor foundation. The pulleys are made from cast steel and lined by replaceable polyurethane tread rims to increase the life of the cables, splices and also to reduce the maintenance cost and noise pollution. The average life of each pulley is around 10000h to 11000 h if properly maintained. Bottom line pulleys are pitched wider as compared to the top line pulleys.

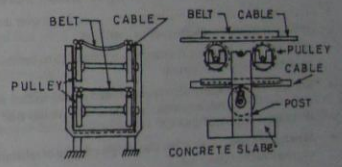


Fig. 7.24 : Schematic Line Diagrams of Line Stand.

The cables passes over two driving surge wheel mounted one on either side of the gearbox unit is driven by a single or two squirrel cage induction motor via a torque converter (or scoop type fluid coupling). In case a slip ring induction motor is used, a hydrostatic motor, a hydraulic braking system and a lubricating pump are employed. The total power of the motors may vary depending on the length, lift and capacity of the cable belt conveyor. However total power of a single flight belt conveyor may vary from 26 kW to more than 7500 kW. The surge wheels are made from cast steel lined with rubber lining or by polyurethane tread.

The cables make 1.5 to 3.5 turns round the surge wheels with head discharge unit and 2 to 4 turns in the case of jib discharge unit.

Belt of the cable belt conveyor is of either single or 2-ply fabric construction since it does not require to transmit any longitudinal driving tension. The belt rests on the ropes with the help of longitudinal shoes molded in rubber whole along top and bottom of each edge of the belt (Fig - 7.25). The shoes have grooves of either semicircular or in 'V' shape. The top and bottom shoes may be offset to reduce the span between the carrying ropes. The belt incorporates square-section galvanized spring steel rods (minimum tensile strength 125 kg/mm^2) positioned transversely across the belt at a pitch around 75 mm which act as beam between the cables to carry load. The standard width of the belts in millimeter are 609, 762, 914, 1067, 1150, 1220, 1372, 1522 and the speed of the belt varies from 1.14 m/s to 4.7 m/s. Depending upon the design, the capacity of the cable belt conveyors varies from 127t to more than 2040 t per hours. Length of a single flight cable belt conveyor may vary from 0.3km to more than 30 km. The belt is tensioned independently with the help of rail mounted tension carriage whereas two track mounted tension bogies are used (one for each of the two ropes) to give tension to the endless cables. In the angle station two endless belts are operated by the station by the same traction ropes which are deflected at the station by means of six horizontal deflection pulleys. The loading side of the belt is lifted up from the ropes by closely spaced belt supporting idlers over a length of 8-9 m.

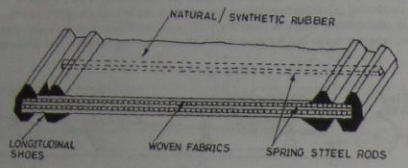


Fig. 7.25 : Schematic Line Diagram of a Section of Belt of Cable Belt Conveyor.

Advantages

A cable belt conveyor has the following main advantages-

- It requires lower capital investment as compared to an ordinary belt conveyor for long distance conveying.
- A gradient up to 25^0 can nicely be negotiated with this conveyor.
- It can negotiate horizontal curves having a minimum radius of 500 m.
- It can handle sticky and abrasive materials which rest over the belt in most static condition and also there is no spillage at all under adverse loading condition.
- Because of less frictional resistance as compared to an ordinary belt conveyor, there is a huge amount of savings of electric power consumption.
- The conveyor can easily negotiate undulated ground surface with less cost as compared to a conventional belt conveyor. Perfect belt alignment is not necessary for this type of belt conveyor. The belt can negotiate a deviation in alignment upto 1.5^0.
- Maintenance cost is less. It has also a higher degree of reliability, flexibility, safety, etc. as compared to a conventional belt conveyor.
- Off centre loading does not affect adversely to the conveyor.
- Changing of belt is easier as compared to a conventional type belt conveyor system.
- It requires less manpower for operation and maintenance.
- It eliminates need of numbers of transfer points as required in the case of a conventional type belt conveyor system for long distance transport.
- Installation cost is less.

Disadvantages

The cable belt conveyors has the following main demerits-

- Since the width of the belt is less as compared to the conventional belt conveyor it has less capacity.
- Belt speed is also very less, which is maximum around 4.7 m/s.
- For drive unit and tensioning arrangement, it requires longer longitudinal excavation in the underground mines.
- It has a very complicated driving and tensioning arrangements.
- In the bump prone mine roadways, this type of belt conveyor is unsuitable.
- Total covering of the belt conveyor is very difficult.

HIGH ANGLE BELT CONVEYOR

One of the difficulties of conveyor haulage is the inherent slope restriction of approximately 18° for handling material which can not negotiate the slope of openpit walls which is more than 45°. A possible solution is a conveyor with a lift capability of approximately 60° of handling high tonnage and high lifts. The HAC offer many advantages over traditional dumper haulage system. These are as follows:

- Less dependency on petroleum products since it is powered by the electrical system,
- Superior energy efficiency-since maximum energy goes for material transport. Only small energy wanted for pulley friction and material acceleration whereas in dumper transport more or less 50% energy goes for hauling tare weight of the vehicle,
- Less maintenance cost and longer life as compared to the dumpers,
- Requirement of operators and maintenance personnel are very less,
- Less haulage road maintenance,
- Excavation is very less, since here ramp etc. are not needed. Only stable slope is sufficient.

Other advantages of high angle belt conveyor includes

- Noise and dust free operation-since material is sealed between the carrying and the conveying belt.
- Easy cleaning and quick repair-belt cleaning by scrapers/plough and repair of the belt is also quicker.
- High conveying angle and high lift-conveying angle up to 60° or more is possible and lift greater than 305 m is possible with steel cord belts.
- Increase in capacity-With the available of belt width 3m wide, conveying capacity of more than 9000 t/h is possible.
- Very simple, high availability, low maintenance cost. etc.
- Flexibility in operation and planning-It is a multi module conveying system using self contained units which may mount on rails, rubber tyres or crawler type transporter or may be equipped with walking mechanism. The extremely anchored high angle belt conveyor may be shortened or lengthened or conveying angle may be altered as per requirement in new location.

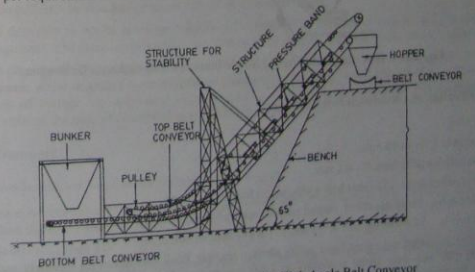


Fig. 7.26 : A Schematic Line Diagram of a High Angle Belt Conveyor

Performance Requirement (General)

- Material characteristics-Angular, hard, slightly abrasive, etc. Angle of repose may be 38°.
- Lump size-Around 20 mm is the maximum lump size.
- Capacity-Average capacity generally more than 3000 tph.
- Lift-Depends upon the machine design.
- Mobility-Less mobility

Types of Sandwich Belt Conveyor

The types of sandwiched belt conveyor are as following:

- The snap sandwich belt
- The mechanically pressured sandwich belt
- The pneumatically pressured sandwich belt

The Sandwich Belt Conveyor Principle

The sandwich belt employs two ordinary rubber belts which sandwich the conveyed material in between. Additional force on the belts provides hugging pressure to the conveyed material in order to develop sufficient friction at the material to the belts and material to material interface so that sliding back will not occur at the design conveying angle (Fig -7.26 and 7.27).

Application

- Deep openpit mining,
- Thin seam strip mining,
- Thick seam mining,
- Multiple thin seam mining with thick parting.

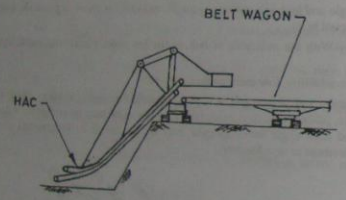


Fig. 7.27 : HAC Suspended from a Single Point is Combined with Beltwagon.

Technical Specification of a Sandwich Belt Conveyor

Overall length: 35m,

Vertical lift: Variable 7.9m to 19.5 m,

Belt speed: Infinitely variable from 0 - 6.1 m/s,

HAC belt : Two endless belts, 1524 mm belt width, nylon fabric reinforced.

Hugging pressure drive : Fully equalized, impact rolls on spring loaded pressure modules.

HAC conveyor drives : 74.6 kW top belt drive and 111.8 kW bottom belt drive

Each drive is shaft mounted onto each head pulley shaft extension. Each consists of an infinitely variable speed D.C. motor 0 - 1800 rpm with over speed capability to 2300 rpm mounted onto a right angle hollow shaft reducer with transmission from motor to reducer input shaft by drive sheave to driven sheave through multiple V-belts.

The nominal reducer ratio is 10:1. There sheaves sets for each drive permit overall reduction ratios of 12:1, 17:1, and 22.5:1.

HAC take ups : Automatic by constant pressure hydraulic cylinder.

: 101.6 mm diameter cylinder at top belt.

: 127 mm diameter cylinder at bottom belt.

: Each take-up consists of the tail pulley on a traveling take-up carriage, a tensioning hydraulic cylinder and wire rope revving to contact the two units.

: A single variable setting hydraulic pump unit provides a constant predetermined pressure to both cylinders.

HAC head drive pulleys : 610 mm overall diameter with 13 mm chevron lagging,

HAC tail take up pulleys: 457 mm overall diameter, mounted on traveling take-up carriage.

HAC luff hoist and revving

Hoisting rope : 19 mm diameter

Hoist-Single drum multiple wrap driven by 7.5 kW, 1200 rpm motor through drive chain and integral worm gear reducer.

Revving- Six part revving between hoisting tower and HAC truss with anchor and hoist at HAC pivot foundation is provided.

Most Common Combinations of Transport Systems

Shovel- Truck Combination

This combination is selected based on the following considerations :

- When haul distance is small (say within 3 km to 4 km)
- When gradient is little bit steeper (best gradient for truck is below 6° however it can negotiate even 12°),
- When size of the ore or overburden rocks are blocky in nature which makes the shovel more suitable as compared to other loader.
- When work is not very much extensive (or patch work), for selective mining and when concentration of work and flexibility of operation is desirable.
- When availability of the access room is very limited and the topography is not suitable for other transport system.
- When requirement of production is from very low to very high (i.e. the production is very much flexible) and life of the mine is short to long .
- When height of the bench is such that shovel is best suitable in that condition as compared to other loader. With this combination both production and productivity of the shovel increase.
- When it is desirable to not to purchase varieties of equipment to reduce the complexity of maintenance and spare parts management. The same combination can effectively be used for removal of overburden rock, soil and ore.
- When it is planned that the quarry will be deep below 200 m and less amount of construction work is to be done. This is because it can negotiate more amount of gradient, less bending radius around 16m to 20 m and require very less amount of surface top alignment. It also reduces much of excavation work and haulage distance; which ultimately reduces the time of construction and lower down the cost.

Shovel-Rail Transport

This combination is selected based on the following circumstances :

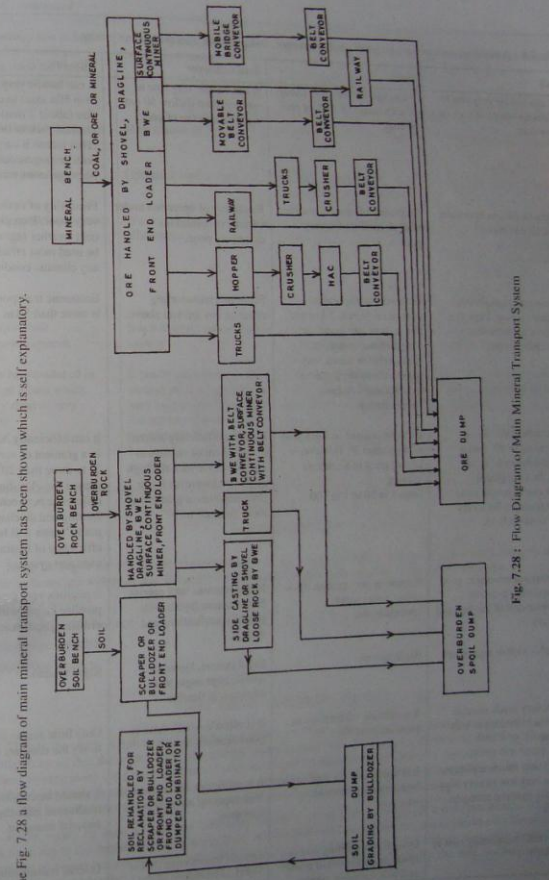
- Haul distance is more (above 4 km). However if the rail network exists very near to the mine, this facility should be used for short distance transportation,
- Gradient is less (less than 3°),
- In a region and unfavorable climatic condition where other transport systems are not very much effective,
- Size of the ore or overburden rocks are large blocky in nature,
- Work is very extensive and flexibility of operation is not much desirable,
- Production requirement is medium to very high and life of the mine is also very long. However if the railway network exists very near to the mine, it can be used for the mine having shorter to medium life with low to medium production too,
- Availability of the access room is quite good;
- When there is a possibilities to invest high amount of money for cheaper material transport system.

Shovel- Trucks-Crusher-Belt Conveyor Combination

This combination is selected based on the following circumstances :

- Haul distance is more (above 4 km).
- Gradient is more and such that the rail transport system is unsuitable.
- Size of the ore is large blocky in nature. The shovel load ore onto the trucks which moves at a distance within 1 km and dump the ore onto the crushing plant. The ore thereafter is crushed to a particular size (below 50 cm) in the crusher to suit the handling by a belt conveyor. The belt conveyor finally transports ore to the required distance and destination. Belt conveyor also negotiates a gradient of around 18° and does not necessitate constructing and maintaining a good haulage road for material transport.
- When it is necessary to connect number of surface mines under the same management to a common dumping point for quality control, reducing the transport, operation and maintenance cost by employing unnecessary huge number of dumpers and their related infra-structures. The system also eliminates the construction of costly major road network particularly in the hilly terrain and supply consistently the required size of ore needed by the processing plant, etc.
- Availability of the access room in the face is very limited where only shovel - dumper match is suitable.
- Requirement of production level is medium to high and the pits have medium to longer life.
- Pit geometry and geotechnical parameters are such that shovel is best suitable under that condition .
- When high investment is possible for material transport system.
- When it is planned that the quarry will be very deep and less amount of construction work is to be carried out. Since it negotiates more gradient (18° or more but up to 16°is the best. With high angle belt conveyor system even a gradient of 60° to 70° can nicely be negotiated), it requires very less amount of excavation work which will reduce the time of site preparation and installation job. The system provides high safety, automation, centralized control, high output, uninterrupted production, shortest transport distance, etc.

A hydraulic shovel unloading on a dumper



In the Fig. 7.28 a flow diagram of main mineral transport system has been shown which is self explanatory.

Fig. 7.28 : Flow Diagram of Main Mineral Transport System

Table 7.8 : A comparison among truck, rail , belt conveyor, and pipe line transportation systems

Truck	Rail	Belt conveyor	Pipe line
1. It can handle very fine to big blocky lumps of rock or ore or coal.	It can handle very fine to big blocky lumps of rock or ore or coal.	It can handle very fine to well crushed (below 50 cm size) rock or ore or coal.	It can handle very fine (less than 50μ size) to optimum size (about 2 mm) well crushed rock or ore or coal .That means it can handle both homogeneous and heterogeneous mixtures.
2. Very flexible in operation under all conditions.	Flexibility of operation under various condition is very poor.	Flexibility of operation under various condition is extremely poor.	Flexibility of operation is very poor. Except in a very cold weather region, it can be used most efficiently in any climatic conditions.
3. Economic transporting distance is about 4 km. It is excellent for medium distance transport.	Economic transporting distance is 4km to 5 km and above. It is very good for long distance transport. If rail network exists very near to the mine, it can be used for short distance transportation.	Economic transporting distance is 4 km and above.	Economic transport distance is more than 5 km.
4. It can be utilized in a gradient up to 10° (but a gradient below 6° is the best). In some places it can also negotiate more steep gradients for a very short distance haul.	It can be utilized in a gradient of maximum 3°. However it is very good in a gradient varying from 1 in 50 to 1 in 100.	It can be efficiently utilized in a gradient of maximum 16°. However with the high angle belt conveyor it can even negotiate a gradient of more than 60° to 70°.	It can efficiently be utilized in a gradient of varying from 2° to more than 20°. The greater the elevation difference between the feeding and discharge points better will be the efficiency of hydraulic transport system.
5. It requires expensive good dry roadways to move, otherwise cost of the tyre will be heavy.	It requires very precise track layout with proper specifications.	It also requires very precise belt structure layout with proper specifications and design.	It requires very precise pipeline layout with proper profile, specification and design.
6. Highly mobile equipment	Rigid system.	Rigid system. However mobile high angle belt conveyor is flexible.	Rigid system.
7. It is very much suitable where accessibility is very limited.	It is suitable where there is good accessibility.	It is suitable where there is good accessibility.	Only little access is required to lay the compact pipelines.
8. It is very much applicable from a very low to very high capacity material transport system.	It is applicable for medium to high capacity material transport.	It is applicable for medium to high capacity material transport.	It should be considered for medium to high capacity material transport.
9. Overall transporting cost is very high.	Overall transporting cost is lower as compared to truck and belt conveyor system.	Overall transporting cost is lower as compared to the truck system.	Overall transporting cost is very low.

Table 7.8 : continued

10. It requires a low to high initial capital investment depending upon the type of trucks to be purchased.	It requires a high initial capital investment.	It requires the highest initial capital investment among all the four systems in case the hauling distance is the same for all.	It requires a high initial capital investment.
11. Maintenance cost is very high.	Maintenance cost is low.	Maintenance cost is very high.	Maintenance cost is low.
12. Operating cost is very high.	Operating cost is very low.	Operating cost is lower as compared to a tuck system.	Operating cost is very low.
13. Watery condition reduces its efficiency.	Watery condition reduces its efficiency.	Watery condition reduces its efficiency.	It is applicable provided large quantity of water is readily available. Watery condition does not have any detrimental affect on its efficiency.
14. It is a discrete transportation system.	It is a discrete transportation system.	It is a continuous transportation system.	It is a continuous transportation system.
15. It can be introduced in any type of mines which have very short to very longer life.	It can be introduced in the surface mine which have a very longer life. However if the railway network exists very near to the mine, it can be used for the mine having shorter to medium life.	It can be introduced when the life of the surface mine is medium to long.	It can be introduced when the life of the surface mine is very long.

ROPE HAULAGES

Rope haulages are one of the cheapest media of mine transportation system. In many small capacity surfaces mines this systems are widely in use. Basically haulages are classified into the followings four categories :

1. Direct rope haulage
2. Main and Tail rope haulage
3. Endless rope haulage
4. Gravity rope haulage

DIRECT ROPE HAULAGE (Fig .7.29)

This is the most simplest type of haulage system having only one track. Gradient for this system should be sufficiently steep so that the empty train gets enough momentum to move freely in the downward direction. Generally if the gradient is very steep , the haulage drum is declutched and allowed to move freely by controlling the brake. But in most of the cases motor is allowed to rotate in the reverse direction. Gradient of the roadways should be not less than 2.86^0 and 4.76^0 for the mine cars having ball/roller bearings and journal bearings respectively.

It can nicely be used to haul one tubs/mine cars to a set of mine cars/tubs of maximum around 25 to 30 in numbers from dip to the required point of the ground. At that place this loaded set of train is shunted by lowering down and a new set of empty train is attached with the haulage rope. The loaded mine cars/tubs are thereafter tippled onto a dumper/railway wagon/bunker one by one either with the help of a tippler or simply by the side discharge type mine cars/tubs itself.

The main units of a direct rope haulage consists of main frame, main shaft assembly, reduction gearing box/system, coupling, service brake, emergency brake, depth indicator, clutch, etc.

Main frame is a welded construction of rolled steel section over which main shaft assembly is installed. Here gearbox and motor units, depth indicator and other control equipment are mounted. The haulage drum consists of a drum shell (split bolted construction) bolted by two cast iron discs in either sides. Hard wooden linings are provided around the periphery of the drum shell. Discs are fitted to the main shaft. In one side brake disc is provided for the brake path. Depth indicator's gearing system is also connected with the main shaft assembly. Drive and intermediate shaft assembly is connected with the babbit - lined journal bearings. Power is transmitted to the haulage drum by a squirrel cage induction motor via coupling (fluid or mechanical coupling) and gear box system. The weight operated service and emergency brakes lined with ferrodo - lining is incorporated with the brake drum and the brake disc mounted on the drum respectively. Direct rope haulages are available in various power line 15 kW /30kW/50kW/65kW/75kW/ 100kW/115kW/ 130kW/150kW, etc.

Rope speed, rope diameter, rope pull, etc. varies from 1m/s to 3.4 m/s, 16mm to 25mm and 1.2t to 4.5 t respectively depending upon the type of the haulages. Capacity of rope in a drum varies from 400m to more than 1200m. Diameter of a drum for a 115kW haulage is around 1.2m having flange diameter 1.9m. RPM of motor is generally kept at 1450.

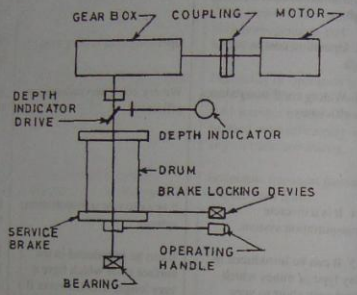


Fig. 7.29 : Direct Rope Haulage

MAIN AND TAIL ROPE HAULAGE (Fig - 7.30)

In this case in the main shaft assembly, there are two drums one for the main rope and the other one for the tail rope. The main rope hauls the set of loaded train where as the tail rope transports back the set of empty train. This type of haulage system is very much useful in the flat and undulating ground surface to negotiate bends/curves and can operate in a single track in narrow path. In view of operation it is very clear that the length of the tail rope is double the length of the main rope. During hauling up, a set of loaded mine car/tubs is attached with the main rope. After clutching the main drum and declutching the tail drum, the train is hauled up. At that time rope is released from the tail drum. If necessary a slight braking pressure is applied to the tail drum braking path for better control and operation. During transporting of the empty mine cars/tubs in the reverse direction, tail drum is clutched and main drum is disengaged. The haulage thereafter is run and light braking is applied if necessary to the main drum for better control and operation. Main and tail rope haulages are available in various power ranges like 30kW/50kW/75kW operating at system voltage ranging from 440 volts to 3300 volts. Rope diameter, pull and speed are 16mm to 22mm, 1.6t to 3 t and 1.5m/s to 2.2m/s respectively. Capacity of haulage drum generally ranges from 500m to 800m length of rope. In surface mining application main and tail rope haulage can nicely be introduced in flat and undulated ground.

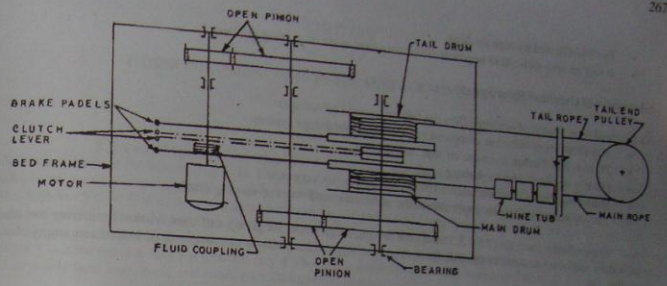


Fig. 7.30 : A Schematic Line Diagram of a Main and Tail Rope Haulage

ENDLESS ROPE HAULAGE (Fig 7.31)

In an endless rope haulage, an endless rope moves continuously round a drive and a return sheave. Specially designed mine cars are attached one by one with the rope with the help of either clipping arrangements or by armoured chains. Two tracks are used with this endless rope haulage system, one is used for hauling the loaded mine cars and the other one is used for transporting back the empty mine cars. Tubs/mine cars may be attached with the moving rope either in single unit or in sets. Pulling of ropes may be arranged either by overhead rope or by an under rope system. In overhead rope system rope moves near the roof whereas in the under rope system rope moves underneath the mine cars/tubs.

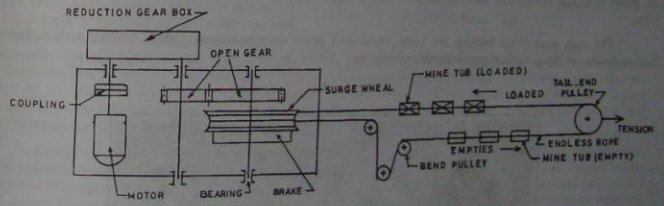


Fig. 7.31 : Endless Rope Haulage

Advantages of Under Rope Haulage System

It has got the following main advantages:

- The tubs/mine cars can fully be loaded and draw bars get direct pull.
- Automatic attachment and detachment of the mine car/tub clip is possible.
- More safe since moving rope is kept at the floor level.
- Curves/bends can easily be negotiated with the help of special guide pulleys.

Disadvantages of the Under Rope Haulage System

It has the following main disadvantages:

- It requires more rollers,
- There is more friction and wear of rope and sleepers.

- Floor is affected by rope slashing.
- It will be very difficult to negotiate undulating road-ways of varying gradient.

Advantages of Overhead Rope System

An overhead rope system of haulage offers the following main advantages:

- It requires fewer rollers as compared to an under rope haulage system.
- There is less friction and wear on rope and sleepers.
- Floor is unaffected by the lashing rope.
- Manual attachment and detachment of mine cars/tubs are very easy.
- Manual attachment and detachment of mine cars/tubs are very easy.
- The method can nicely be applicable in the undulating road-ways of varying gradient.

The endless rope coils (2.5 - 3.5 turns) the driving surge wheel for giving sufficient friction for driving and also to provide a sufficient amount of tractive force. The system provides continuous movement of loaded and empty mine cars/tubs.

The endless rope haulage system as a whole provides the following main advantages-

- System is well balanced.
- Power cost is lower as compared to a direct rope haulage.
- Can be nicely applied to negotiate undulating roadways of varying gradient.

But the system has the following demerits:

- Whenever a breakdown occurs, the total system comes to a standstill.
- More manpower is needed in this system.

Power of the endless rope haulage is generally 22.5kW/30 kW /37 kW /50 kW /75 kW /100 kW having system voltage ranging from 440 volts to 3300volts. Diameter of rope, rope speed and maximum static unbalance rope tension are 19mm to 28mm, 0.7m/s to 1 m/s and 3t to 7 t respectively. Diameter of surge wheel in 50 kW haulage is around 1.2m.

The main units of the haulage are frame, main shaft assembly, transmission system, braking system, brake locking device, coupling, squirrel cage induction motors, etc.

Front end loader

CHAPTER 8

OTHER EQUIPMENT USED IN THE SURFACE MINING SYSTEM

SCRAPER

It is a diesel operated four wheel drive rubber tyred tractor or a crawler tractor having a bowl attached with a cutting blade at the bottom (Fig.-8.1). The scraper can cut a thin slice of unconsolidated material ,soft rock or soil (thickness of the slice is approximately varying from 70mm to 250 mm) and can nicely be used within a radius varying from 150m to 1500 m. Size of the material which it can handle varies from very fine to more than 550 mm in good and dry ground condition. However wet ground condition hampers the scraper efficiency. It is very much flexible under wide varied conditions and can handle wide rate of production ranging from very small to very high total tonnage. Maximum efficiency is achieved at a very flat gradient (say 4^0) but it can negotiate a gradient even more than 15^0.Scraper cut and load or simply load, haul, dump and return back for recycling of operation.

Scraper can also be used for hauling blast fragmented hard rocks or minerals. Scraper should be used for removal of top soil or unconsolidated rocks or minerals of thickness 2m to 3 m. The speed of a tractor wheel mounted scraper is around 60km/h to 65 km/h. Because of higher mobility the wheel mounted scraper is preferred over the crawler track mounted scraper. However where steep gradient, seasonal variations, etc., are to be encountered and where mobility is not a factor of selection , the crawler track mounted tractor will yield a very good result. The tractive effort of the scraper depends on power, weight of the machine, gear efficiency, type and size of the tyres, condition of the surface topography over which it moves, etc. During cutting most of the times a bulldozer is necessary to push the scraper from the backside. The scraper may be used over a ripped or hard mineral or rock surface for loading and transporting purposes. It is also used for leveling purpose. However it requires a very good traveling ways. It is unsuitable in hard ground condition and sites having big boulders. The scrapers are basically divided into two categories i.e. (1) Self propelled type and (2) Towed type.

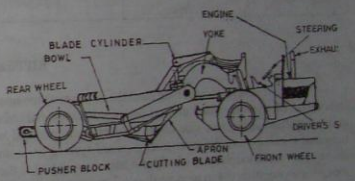


Fig. 8.1 : Scraper

Rubber tyred self propelled scrapers are available as four wheel units or as two wheel overhung units. Because of the stability, smoothness of movement, ease of operation, higher speed (around 45 km/h to 50 km/h), higher safety, four wheeled scrapers are widely used. Of course sometimes due to lower investment, lower operating and maintenance cost, greater degree of maneuverability two wheeled overhung scrapers are also selected. The pusher tractor or bulldozer provides the extra tractive effort in it during cutting and loading operation.

The towed scrapers are mounted over four wheeled machine body and attached to the rear side of the tractor. A scraper consists of the following main units and components: bowl, apron, cutting blade, steering system, power transmission system, cable or hydraulic type of control unit, braking system etc.

After retracting the tailgate to the extreme position, the apron is opened partially and the bowl is lowered till the cutting blade touches or penetrates in the loose ground with high hydraulic pressure. The scraper thereafter moves forward taking a slice of 100 mm to 200 mm of the soft, unconsolidated and loose ground material which fed into the bowl till it is filled up. Thereafter bowl is raised up and the excavated material is hauled and dumped to the required place.

Whilst operating on a bench, a scraper should not approach within 2 m of the bench crest. No scraper should be permitted to move backwards downhill when unloading.

Scrapers hauled by wheeled tractors should not be permitted to negotiate access roads having a gradient greater than 15^0 in the case of a loaded machine or 25^0 in the case of an empty machine.

The available struck capacity of a scraper may be more than 12 m^3 whereas the heaped capacity of more than 16m^3 is also possible. Table-8.1 gives specification of some of the Wheel Tractor Scrapers.

Table -8.1 Wheel tractor scrapers

Make	Model	Flywheel power (kW)	Operated weight (t)	Scraper struck (m^3)	Capacity heaped (m^3)	Top speed (Loaded) km/h	Width of cut (m)	Max. depth of cut (m)	Max. depth of spread (m)
Caterpillar	621E	246	30.4	10.7	15.3	51	3.02	0.33	0.52
Caterpillar	631E	336	44	16.1	23.7	48	3.5	0.437	0.48
Caterpillar	651E	410	59.42	24.5	33.6	56	3.83	0.44	0.66
Caterpillar	627E	Tractor-168 Scraper-168	34.67	10.7	15.3	55	3.02	0.33	0.52
Caterpillar	637E	Tractor-336 Scraper-187	50.84	16.1	23.7	48	3.5	0.437	0.48
Caterpillar	657E	Tractor-410 Scraper-298	68.72	24.5	33.6	55	3.68	0.44	0.66

RIPPER

It is a farmer's plough type steel shank (furrow) mounted with cutting tooth and attached with a steel beam at an interval of 1m to more than 2 m apart (average total numbers of shanks are 4 but 6 teeth are also available) and the whole unit is attached at the rear side of the crawler track mounted heavy duty diesel operated tractor (in case of the tractor mounted type ripper). However tractor pulled type ripper comprising of a draw tongue and a frame attached with a shank. The steel plough body attachment has a cutting tool at the bottom of it which dips into the ground (to be ripped) at a depth varying from 0.4 m to more than 1.2 m depending upon the design of the machine. It is done by the thrust applied to the hydraulic system. When the tractor starts moving the friable soft to medium hard rock or mineral body breaks properly which are loaded thereafter with the help of a scraper or a loader or simply dozed with the help of bulldozer. This type of machine is suitable for ripping alluvial surface, soft rock, medium hard well stratified rock, weathered sand stone or shale type rock, soft to medium hard lime stone, laterite deposits, coal, well fractured hematite iron ore, etc. The degree of rippability depends upon the brittleness of rock, degree of stratification and lamination of rock, well defined fracture plane, moisture content in the rock, geological disturbances like fault and other fractures, grain size, degree of consolidation and weathering, wet condition physico mechanical properties of rocks like compressive strength, tensile strength, shear strength, etc. Sometimes pre-fracturing of the consolidated ground (by blasting) is necessary for efficient ripping operation. The rippability of rocks can be determined by measuring the magnitude of the seismic wave velocity in the rock mass. Lower the velocity better would be the rippability of rock. A seismic wave velocity less than 3000 m/s in the rock are amenable for ripping. The ripping (scarification) depends upon the nature of the ground, power and type of the ripper,m the downward thrust to the shank tooth, weight of the tractor unit, etc.

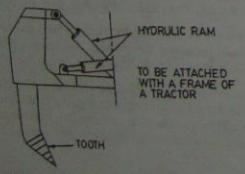


Fig. 8.2 : Schematic Line Diagram of a Portion of a Tractor Mounted Ripper

Ripper (Fig.-8.2) cutting teeth is forced to penetrate into the ground with the help of hydraulic pressure and cause a shear failure of the rock when the force exceeds the compressive strength of the rock mass. The tractor when

starts moving the drawbar pull causes tensile breakage of rocks. In case of fractured rockmass, the rock failures occur when the drawbar pull exceeds the cohesion between the fractured blocks of rocks. The economical speed of the ripper is around 2 km/h to 3 km/h when maximum drawbar pull is available. A speed more than that amount will cause consumption of more power along with excessive track slippage, wear in the undercarriage unit and the cutting teeth, etc. It will also consume more power, generate excessive heat and will cause low production. The ripping depth depends upon the thickness of the lamination, degree of hardness of material and fracturing, efficiency and condition of the machines particularly the cutting teeth, tractive force of the machine, etc. More the ripping depth means more rate of production at the lesser operating cost. It is always better if the depth of penetration equals to the depth of the lamination of the strata. At that condition the requirement of motor power will be less. However the depth of the ripper should be such that the driving sprockets should not touch the ripping ground. The spacing between the teeth mounted on the steel shank depends upon the use of mineral or rock, the size and type of the loading equipment, size and type of the crusher, size and type of the transporting equipment, tractive effort of the equipment, fracturing characteristics of the ripped rock or mineral body. If the spacing between the two shanks is more, there will be the formation of coarser materials. Hence the spacing should be the optimum. The direction of ripping depends upon the presence of vertical laminations or fracture plane (ripping should be across the vertical laminations for obtaining proper fracturing otherwise along the parallel line ripping will produce only deep channels); direction of ripper and scraper movement (ripping should be done along the movement of the scraper and towards the downhill side to get maximum scraper's capacity and ripper's power efficiency respectively), etc. If the materials are very hard ,widely laminated or massive in nature , cross ripping (ripping at a particular area is done first ripping into longitudinal passes and then ripping the same area by the transverse passes. The first ripping will loosen the vertical lamination and will produce enough fracture planes so that the second pass easily rip that part of the total area), tandem ripping (here ripper is assisted by another tractor which pushes the ripper for increasing the drawbar pull to negotiate the hard rock formations) or pre-blasting with the help of light charge are to be practiced.

Various types of rippers which are available in the market are as following-

- Hinge type ripper-In this type the linkage hinge carries fixed position at the rear end of the ripper. Various differential movements of the ripper teeth in the vertical direction due to the up and down movement of the ripper cause the change of the teeth angle to meet the varying rock conditions.
- Parallelogram type ripper-In this type of ripper the linkage carrying the beam and the shank has the constant teeth angle regardless of the teeth depth and have excellent penetration characteristics. The ground clearance between the tracks and the shank of the ripper and the raising height of the ripper is more which enable to rip more blocky lumps and ease out the checking of shank tip respectively.
- Impact ripper-In this type of ripper the engine power is converted into hydraulic power to impact rapidly to a specially made ripper shank. The energy obtained from those impact pulses transmitted into the ripper tip causing heavy fracturing and increasing the penetration depth into the rock or mineral. It also reduces the draw bar pull. With this type of machine, ripping of hardest rippable rock is possible.

BULLDOZER

It is a diesel operated tractor mounted machine having a dozer (pusher) blade attached to the front of it. The machine is either wheel or crawler track mounted having hydraulic arrangements to raise the blade up and down. The concave straight blade's mould board is attached with cutting edge to dig into the soft rock or soil surface and doze the loose cut rock or mineral body (Fig.-8.3). The machine is also used for land preparation, cleaning, construction and maintenance of haulage roads, benches etc. It also serves as auxiliary back up service machine for piling up the blasted rock mass for shovel and dragline, bulldozing the waste pile, making proper slope in the bench, cleaning up of bushes, grabbing, salvaging and shifting of machines and materials like switches, cables, pipes, pumps, skid mounted various auxiliary machine units, etc. They are also used for pulling heavy vehicles into the workshop for proper repair and maintenance when they are under breakdown for pushing coal into ground bunker through grizzly for reclamation of land, leveling, spreading and compacting of material in the dumping yard etc.

The bulldozer is most suitable equipment to handle the rock mass under the following conditions-

- To doze fine to large range blocky rocks or minerals,
- To work within the operating radius of 150 m,

- In all ground conditions (rough terrain, soft or wet ground)
- It can also work at a gradient of 13° to 14°, (however the wheel mounted rubber tyred vehicle can negotiate only in the mild to moderate gradient), it is very much flexible under various ground condition and has a high degree of mobility (however mobility in case of crawler mounted bulldozer is less),.
- If the machine is used for hauling purpose it's production rate is very low.
- Dozers are also used for relocation of stockpile, spreading waste material on dumps, dozing of supplementary materials, pioneering and maintenance of roadways, etc.

During selection of a dozer the following criteria shall be considered:

- Hourly production requirement
- Number and size of equipment required
- Attachment required (e.g. ripper) for special purposes
- Crawler track or rubber tyre mounted machines
- Suitability of machine in various topographic conditions
- Site specific conditions and environment
- Operating method
- Requirement standard of maintenance
- Requirement of skill of the operator
- Performance statistics of the equipment in the other mines
- After sales service
- Availability of indigenous equipment and spare parts.

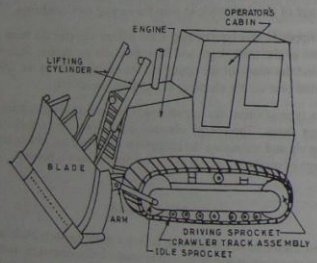


Fig. 8.3 : A Schematic Line Diagram of a Dozer

Safety during Operation

When operating a tractor-type bulldozer, it should be prohibited to leave a machine with the engine running and the blade lifted; stand on the blade frame or on the blade; operate a machine that is not fitted with an interlocking device to prevent the engine from starting when it is in gear; and operate a machine that is not fitted with a device which requires the engine to be started from the cab.

All repairs, lubrication or adjustments to a bulldozer should be performed with the machine on level ground after switching off the engine and the blade lowered to the ground. Where, because of failure or defect, a bulldozer is stopped on a slope, steps should be taken to block the machine in position so as to avoid any downhill movement before repairs are attempted.

Should it be necessary to carry out an inspection of the underside of a bulldozer blade, the blade should be lowered on to firm supports and the engine to be stopped. No person should be permitted to go under a lifted blade except it is firmly chocked up and permission has been given by the competent person.

Bulldozers should not be permitted to operate on slopes where the inclination measured along the steepest plane exceeds 25° or on any slope which has a dangerous cross inclination.

The blades which are generally used with the dozer are generally light to heavy in weight depending upon the condition of rocks. For hard rock excavation the blade might be attached with cutting teeth whereas during grubbing rake blades are used. It is already mentioned that in conventional type of bulldozer the blade is kept in right angle to the longitudinal axis of the tractor whereas in angle dozer the blade can be turned at an angle to the direction of travel to cast the loose rock, soil or well fragmented rock to either side of the tractor when the tractor moves forward. The angle of blade is generally fixed but in some cases it may be varied. A dozer can dig more than 1.5m below the ground. The blade factor in case of easy dozing, average dozing and difficult dozing conditions are 0.9 to 1.0, 0.75 to 0.9 and 0.35 to 0.75 respectively.

Operating weight and drawbar pull of bulldozer may be possible in the range varying from 35 t to more than 45 t and 65t to more than 90 t respectively. Power and traveling speed may be possible in the range varying from 67.5 kW to more than 525 kW and 12km/h to 13 km/h respectively.

An example

Komatsu's D275AX-5 crawler dozer features a new K-bogie-design undercarriage to help the unit hug the terrain for better contact and maximum drawbar pull. It incorporates a hydrostatic steering system powered by an independent hydraulic pump that transmits engine power to both tracks without an interruption on the inside track. Steering and blade-control systems have a palm command control system. Komatsu's Tier II turbocharged, after cooled engine delivers 307kW at 2,000 rpm. It's weight is around 50 Tonne.

CLAMSHELLS

It consists of a long boom like a dragline having two sheaves (one for hoist line and the other for digging line) through which hoist and digging rope pass and connects to a specially made clamshell bucket. The bucket consists two movable jaws (hinged as a movable rod in one end and on a brackets in the other end) having curved lips. The hoist rope is connected to the head beam by a socket. The bracket and digging rope are also connected to the head beam. After releasing the digging line, the main shaft is lowered down to a proper position by the action of gravity. The bucket opens out and digs into the ground by its own weight. When the hoist rope is revved, the jaw moves by scraping the materials and afterward grabs them. Jaws of the bucket close and the bucket is lifted up along with the trapped material (Fig.-8.4). The bucket is generally heavy in weight in case of hard digging material and lighter in weight in case of loose and easy digging material. The bucket may have teeth to collect very well crushed rocks or ore. But they may be teeth less if loose sand under water material, etc., are to be excavated. For medium size blocky ore or rock, a cactus grab and for excessive large boulders an orange peel grab can ideally be used. A clamshell has longer boom to cover a wide radius of operation. Clamshells are very much suitable for vertical lifting of loose to coarse materials from bottom to top level or on to the surface. The machine after lifting swing the bucket to the proper dumping spot where the digging line is loosen to open up the bucket to discharge the material over a haulage unit or above it's level. It can nicely be used to rehandle spoil dump, to load crushed ore, coal or other mineral from the stockpile to the haulage unit, to remove muck from the bottom of the sinking shaft, to remove mud from the excavation under water trenches and large foundation work, etc. It can also be used to handle big rock or ore boulders, stumps, etc. Its cycle of operation is very high and output, efficiency etc. are very poor as compared to an identical shovel, backhoe, dragline, etc.

Operation

The operation of a clamshell follows the following broad steps:

- Position and level the equipment , ensuring the digging operation is as close to the radius as the dumping operation.
- Select the correct size and type of bucket
- During lowering the clamshell bucket, if too much pressure is applied to the closing line brake, the bucket will close and an excess amount of wire rope will unwind from the holding line hoist drum. To avoid this,

release the holding line and closing line brakes simultaneously during lowering the open clamshell into the material for the initial bite. Engage the closing line control lever to close the bucket. Control the digging depth by using the holding line control lever and brake.

- If, during hoisting, the hoist line gets ahead of the closing line, the bucket will open and spill the material. (This could also be caused by having too much wire rope on the hoist drum.) The operator must hoist both the closing and holding lines at the same speed to keep the bucket from opening and spilling material.
- When the clamshell bucket is raised enough to clear all obstacles, start the swing by engaging the swing control lever. Hoisting the bucket can be performed, as it is swung to the dumping site. The spring-loaded tag line will retard the twisting motion of the bucket if the swing is performed smoothly.
- Dumping and unloading the clamshell is performed by keeping the holding line brake applied while the closing line brake is released. Apply the closing line brake quickly after the load is dumped to prevent the closing line from unwinding more wire rope than is needed to dump the material. After the bucket is emptied, swing the open clamshell back to the digging site. Thereafter lower the open bucket and repeat the cycle.

The clamshell operating cycle has four steps: filling (closing) the bucket, raising the loaded bucket, swinging and dumping. The boom angle for clamshell operations should be between 40° to 60° . Be careful when working with higher boom angles, as the bucket could hit the boom. A clamshell attachment is not a positive digging tool. The height reached by the clamshell depends on the length of the boom used. The depth reached by the clamshell is limited by the length of wire rope that the hoist drum can handle.

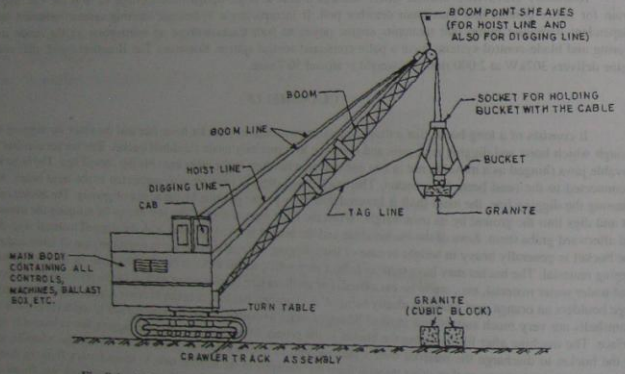


Fig. 8.4 : A Generalised Schematic Line Diagram of a Clamshell Under Operation

MOTOR GRADER

It is a pneumatic wheel driven equipment used mostly for grading and leveling surface land, light cutting and planning of bench floor and haulage road, cleaning of spillage rocks or ore boulders over the haulage road, etc. Although it is a lighter machine but quite longer in shape and basically bifurcated in two portions.

The back portion is the main body mounted over four larger pneumatic tyre wheel and all the control and driving arrangements are incorporated in it. The front portion is mounted by two smaller pneumatic tyre wheels and connected with the rear main body by a cross braced frame [Fig. 8.5 and 8.5(A)]. A circle having gear teeth is suspended from the cross braced frame. A grading blade (around 4 m approx) is attached with the circle which can rotate to change the blade angle depending upon the condition. The blade can be pulled by a drawbar attached in the front side of the

body. A side shifting arrangement control the movement of the circle along with the blade and the later is held in position with the help of a spring loaded lug. The bottom edge and sides of the blade are replaceable type. When these edges damage, they are promptly replaced. The grader is moved forward while in operation but it can also be reversed or the direction of blade may be reversed for doing spreading or smoothening job. The blade can be tilted in any side (around 20° to 25°) or lifted up or can be tilted vertically whole of it sufficiently to dig the floor. It can also be shifted to any side to a considerable length. If any obstruction created by any boulders or hard materials is to be overcome ,the blade is simply lifted up. Amotor grader works for side casting to push the material along the side by regulating blade angle. planning by sufficiently digging, cutting , spreading and finally filling the hollows, shaping of road being constructed newly by digging ditches and using soil to make the bed of the road, crowing the road with stone chips, or small gravel or by dirt ,etc. A motor grader should never be used for a site having less than 300 m in length and width less than 12 m since turning of the grader is very tedious task (because of its longer size and shape) and results much of scuffing and gauging of the ground surface. Sometimes scarifier is attached in front of the grader blade to rip the hard rocks to ease the operation of cutting and grading. In same of version provisions are kept to attach dozer blade in front of the rear main body to doze more materials. The working capacity of a grader depends on the tractive effort and power of the machine. Grader may be available in various sizes, models and capacity. Its power ranges from 25kW to 205 kW and weight ranges from 5 t to 25 t. Table 8.2 and 8.3 gives specifications of some of the motor graders used in the opencast mines.

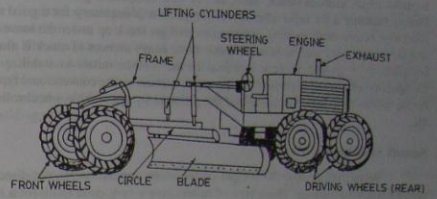


Fig. 8.5 : A Schematic Line Diagram of Self Propelled Motor Grader

Table : 8.2 Motor grader for opencast mines.

Make	Model	Flywheel power (kW)	Operating weight (t)	Rated engine RPM	Top speed		Standard blade		Lift above ground (m)
					Forward (km/h)	Reverse (km/h)	Length (m)	Height (m)	
Caterpillar	120B	93	12	2000	35.4	23.8	3.66	0.61	0.38
Caterpillar	140B	112	13.6	2000	37.6	25.6	3.96	0.70	0.35
Caterpillar	120G	93	11.48	2200	40.9	40.9	3.66	0.61	0.45
Caterpillar	130G	101	12.27	2200	39.4	39.4	3.66	0.61	0.47
Caterpillar	12G	101	13.31	2200	39.4	39.4	3.66	0.61	0.47
Caterpillar	140G	112	13.52	2200	41.0	41.0	3.66	0.61	0.47
Caterpillar	14G	149	18.53	2000	43.0	50.1	4.27	0.68	0.48
Caterpillar	16G	205	24.73	2000	43.6.	42.6	4.88	0.79	0.44

Table : 8.3 Various types of motor grader

Manufacturer	Case	Caterpillar	Galion	Komatsu	Lee Boy	Volvo
Model	885	140H Updated	870 Series C	GD825A-2E	685	G780B
Frame type	Artic	Artic	Artic	Artic	Artic	Artic
Operating weight, Tonne	17	14	15	26	7	19
Moldboard length (m)	4.2	3.6	3.6	4.85	3	4.2
Net engine power, 1st gear (kW)	154	124	153	210	82.5	164
Variable power, net, max (kW)	-	154	-	-	-	182
Front wheel drive	6x4 std.	6 x 4 std	6 x 6 opt	6 x 4 std	6 x 6 opt	6 x 4 std

COMPACTING EQUIPMENT

Compaction of ground is a process of pressing the material of the ground due to which the density and strength of the material increase. Pressing will remove air and water present in the pores of the materials and hence permeability of the compacted materials will decrease. The compaction stabilizes the ground of artificially built up fills which is done to make roads, embankments, dams, etc. A well compacted ground does not affect very much by rain or bad weather. During building a fill layer after layer, compaction is necessary for a good compaction and stabilization of the ground, otherwise heavy earth moving machineries will get stuck up under the loose soil and also power requirement to run any mobile machineries will be heavy. Beside these heavy amount of muck is also produced on loose soil by the rain. Hence compaction of ground is very essential in the surface mines to stabilize and compact the roads, bench floor, etc. Compaction of ground is possible by pressure applied by the conventional type steel wheel rollers, vibration and pressure by the vibratory rollers, kneading and the pressure by the wobble-wheel rollers, impact by the rammer type equipment, etc.

Smooth Steel Rollers (Fig-8.6)

When rollers run, the weight of the same applied across the direction of its travel and produces a straight, smooth and solid surface. These type of rollers give a very good result for a layer of loose soil or well fragmented rocks of thickness varying from 100mm to more than 250 mm. Generally good squeezing is resulted on low spot material spreads over the road. These types of rollers are very much useful to compact road surfaces, pavements, fills and sub-grading. The three wheels and tandem model consists of a pair of large diameter rolls in the rear and a smaller diameter but wider steering rolls in the front. Weight of the smooth steel rollers (three wheel tandem model) may be around 2t to more than 20 t but weight of the roller can be increased by inserting sand inside the rolls. Two rolls tandem rollers are also available. The rollers may also be equipped with water sprinkling system and scraper blades. In three axle tandem rollers there are three axle over which the main body is placed. The front axle along with front roll is connected with a walking beam which can be locked and unlocked depending upon the condition of the ground profile. If the walking beam is locked , the front weight will be concentrated both in the front and in the middle rolls while negotiating an undulating profile. If walking beam is partially locked, then during negotiating undulating profile, the front roll rises up but the front weight will be concentrated on the middle roll to compact the ground. When the walking beam is unlocked the weight of the rollers will be distributed equally on the three axle.

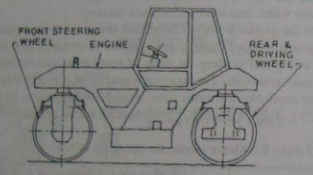


Fig. 8.6 : Smooth Steel Roller

Main Components of the Roller

Frame

The rigid frame consisting of thick steel sheets is manufactured as a welded construction and is of the same design for all types. The front and rear modules are bolted to the frame and can therefore be easily dismantled. The driver's cock pit is located in the middle of the roller and is designed according to ergonomic norms. All the instruments and controls are arranged clearly and are within easy reach. The shock absorbing body-molded seat can be adjusted according to the driver's height and weight and can also move laterally. So the driver can so alter his seating position as to ensure a clear view of the roller edges.

Engine

Air or water cooled four-cylinder diesel engine which provides sufficient to performance reserves for all operating conditions, with low levels of fuel consumption.

Drive

It consists of hydrostatic rear or all wheel drive (depending on type) unit. The transmission system consists of an axial piston pump driven by the diesel motor or wheel hub motors having flange mounted on the driven drums and provides hydraulic power transmission. With all wheel driven models, the differentials compensative between front and

rear drum is provided with hydrostatic system. To improve the traction, a hydraulic differential lock can be applied. The entyre hydrostatic drive system is controlled by a single lever which allows smooth and gradual changes in speed and direction and also simultaneously functions as an operating brake. The roller is secured against involuntary starts by a neutral start device and a safety lock or the driven lever.

Steering

It consists of hydrostatic front or all wheel steering (depending on type) system. The oil pressure, which is generated by a separate hydraulic pump, is transferred to one hydraulic steering cylinder per steered axle via a control valve and is activated by the steering wheel. The auxiliary steering provides exact control at a low power input. In the case of a failure in the hydraulic support system the roller can still be steered. With all wheel-steered at the same time, makes the roller highly maneuverable and allows operation with off set drums. Thus the available compaction width can considerably be increased on requirement and exact rolling up to border and edges is possible. In addition, off set steering offers extra safety against lateral sliding or tilting.

Vibration

Vibration is initiated by hydrostatically-driven vibrators on both the drums. The fluid in the hydraulic motors installed in the drums is supplied individually by separate oil circuit. The oscillation frequency can be adjusted without graduations between 35 - 50 Hz, the amplitude can be chosen independently of frequency in two stages (0.2 mm and 0.45 mm). The vibrators can also be operated independently of each other at high or low amplitude. The vibrator is switched off before each change of direction to avoid the formation of depressions in the road surface.

Brakes

Hydrostatic brakes act on all driven drums. The hydrostatic drive functions simultaneously as an operational brake thereby rendering a separate control superfluous. Parking brakes also act on all driven drums. The parking brake is operated by a spring accumulator and can only be released by oil pressure from the drive system. When the motor is not running or in the case of a failure in the drive hydraulics, the parking brake functions automatically and therefore serves as additional safety brake.

Sprinkler System

The drums of all production types are equipped with time controlled pressure sprinklers. The water in the large reservoir is distributed by air pressure, produced by a compressor via flat spray nozzles over the whole width of the drum. An interval switch and the sponge rubber rolls mounted on the drums additionally provide economic and continuous moistening. In addition to the middle reservoir, the drum can be filled with water. The whole water reserve is sufficient for at least 12 hours of operation.

Special Equipment

Front light, reversing light, hazard warning light, rotating flash light (yellow), working light, sprinkler tank gauge, water pump, blowing equipment and sponge rubber rolls for sprinklers, pressing device, driven cabin, wind screen washer, heating etc are equipped in the machine. Table 8.4 gives specifications of some of the rollers (compacting equipment).

Sheepsfoot Rollers

In this type of rollers projected feet are mounted over the drum and the drum connected with a box type frame. The box frame is towed to a tractor. The roller on movement rotates and compacts the ground surface with their feet. Penetration of feet over the loose soil is very deep in initial stage but as the roller moves number of times over the same area the ground becomes compact, its density increases and the roller moves out without any further penetration. These types of rollers are very much useful for the fine plastic ground. Feet may be tapered having circular or square faces. The length of the each foot may be around 180 mm to 240 mm.

Table 8.4 Roller (compacting equipment)

Tandem Roller RT-7 , Make Voest Alpine	
Technical specification	7 t
Operating weight	25da N/cm
Linear pressure	0.2mm/0.45mm
Vibration frequency	1.45m
Drum width	
Working width (maximum)	2.5m
All wheel steering	1.45m
Single steering	0.11km/h
Speed forward/reverse	
Inner turning radius	4.5m
All wheel steering	9.5m
Single steering	
Engine options	55kW
Deutz F4L/912 output bace to DIN6270	2500rpm
Rated speed	57kW
Parking 4236 output bace to DIN6270	2500rpm
Rated speed	120 l
Fuel tank capacity	1400 l
Water reservoir	
COMBI rollers RC6	
Technical data	6t
Operating weight ace++CECE Standard	21da N/cm
Linear pressure	30-50Hz .amplitude 0.2mm/0.45mm
Vibration frequency	1.45m
Drum width	
Working width (maximum)	2.45m
All wheel steering	1.45m
Single steering	0.11km/h
Speed forward/reverse	
Inner turning radius	4.5m
All wheel steering	9.5m
Single steering	
Engine options	55kW
Deutz F4L/912 output bace to DIN6270	2500rpm
Rated speed	57kW
Parking 4236 output Bace to DIN6270	2500rpm
Rated speed	120 l
Fuel tank capacity	1100 l
Water reservoir	110-200
Tyre size	

Safety Attachment

The safety attachments incorporated in the compactor, motor grader, wheel tractor - scrapers, front end loaders are as follows:

Role over protective structure, falling objects protections, sound and vibration suppression, seat belt, back-up alarm, warning-horn. rear view mirror, radiator core protection grid, fan blast deflector, heavy duty radiator guard. engine enclosure, track roller guard in case of crawler track mounted vehicle, wind shield wiper and washer, fire extinguisher, spark arrestor, lighting system, crank case guard, etc.

HYDRAULIC HAMMER

The most convenient application in opencast mining for secondary breaking of boulders is a hammer mounted on a regular hydraulic excavator on wheels (Fig-8.7). The structure of a modern hydraulic hammer is very simple. There is a piston moving up and down and striking directly against the tool. To produce big energy pulses during the downward stroke, the hammer is equipped with a nitrogen accumulator which is able to supply the necessary volume of compressed oil in a very short time. The accumulator is charged continuously by the pump of the excavator or by an electrically driven hydraulic power pack. When the quarry equipment is coupled with a hydraulic hammer unit, many advantages will be achieved which are as follows -

- Drilling, loading and haulage can be carried out without interruption and production will increase.
- Fly rocks from secondary blasts do not damage the neighboring machinery, cables and structures.
- Heavy hydraulic hammer can be used in OCM not only for boulder breaking but also for the following purposes:

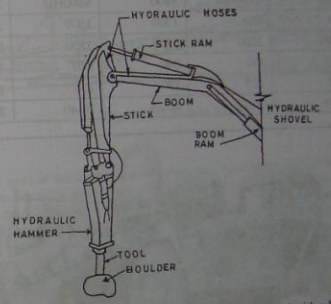


Fig. 8.7 : A Schematic Line Diagram of a Hydraulic Hammer Attached with a Hydraulic Shovel

(i) When a heading blast has failed and the rock pile is so tight that cannot be loaded, the hammer is an excellent tool to loosen the pile.

(ii) In sections where the rock is very fissured, drilling and charging of blast hole is difficult, the big hammer can be used for benching.

(iii) Toes on the quarry floor can be easily removed and faces can be cleaned by the use of hammer.

Weight of hammer, (kg)	Hammer energy (joule)	Boulder size (m^3)	Breaking capacity (m^3/8h)
800	2000	<1	150-250
1600	4000	<3	200-400
2400	6000	<6	300-600
3200	8000	<6	300-1000

For hammer application at the crushing plant Rammer has designed a complete range of pedestal mounted breaker boom-

Boom Type	Hammer used
BBS	S700 and S700 HD
BBM	S800 and S800 HD
BBL	S300 HD and S 1600 HD

Table 8.5 gives the specification of some of the hydraulic hammer manufactured by the Rammer.

For Jaw Crusher the boulder breaking takes place on the feeder and for gyratory crushers the hammer boom can be mounted directly on the feed bin.

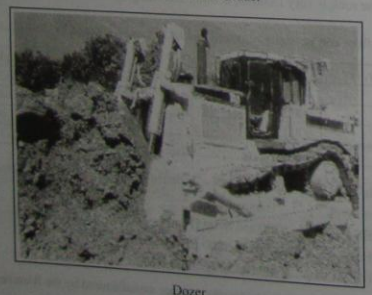
In most cases there is a grizzly placed on the rock pan or on the feed hopper. The only special features which have to be taken into consideration are that the grizzly bar has to be strong enough against horizontal forces. When hammering an oversized boulder through the grizzly, high horizontal forces will be generated. If properly designed, a grizzly may last years without any essential repair. A pedestal mounted hammer when installed at a crusher offers following advantages-

- Crusher output will increase.
- Explosives are not used in the crusher.
- Worn out crusher plates and other parts can be changed by use of the hammer.

Table :8.5 Rammer hydraulic hammer range

Type	S700	S700HD	S800	S800HD	S1600HD	S200HD
Weight ,kg	740	850	1300	1500	2400	3200
Impact energy, j	2200	2200	3500	3500	6000	8200
Impulses Ns	440	440	840	840	1320	2000
Tool diameter, mm	115	115	130	130	155	170
Carrier, t	8-16	16-22	16-25	20-40	25-50	35-80

Fig. 8.5 (A) : Motor Grader



Dozer

CHAPTER 9

STACKER, RECLAIMER AND SPREADER

STACKER

The stacker system consists of a stacker mounted on 10 number crawler track assemblies, a connecting bridge and a belt tripper with built in conveyor drive system for the dump conveyor to handle overburden/ore (Fig. 9.1).

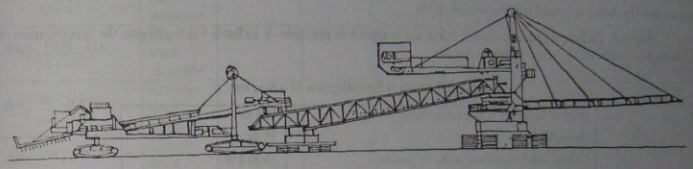


Fig. 9.1 : A Stacker System

Revolving Structure

The structural steel work of the stacker and belt tripper is predominantly of full web and box girder construction for offering best resistance to corrosion and ingress of dirt. The connection between the revolving superstructure and the stacker gantry takes the form of 2 row large diameter ball slew bearing. The contact faces of the gantry and revolving superstructure are machined to warrant a positive seat. The slew bearing is secured by friction grip bolts to the internal ring of the gantry. The ball and slew bearing is lubricated by means of an automatic centralized lubrication system. The ring gear is sprayed with grease. The slew bearing is protected by an additional metal cover with rubber skirt. The fine spur/planetary gearbox of the slew motion are self contained units and driven by squirrel cage induction motors. The power is transmitted by overhung pinions to the ring gear located separately on the gantry. After swing out of the slew drive unit, it can be inspected on the machine without the need to interruption of operation.

Conveyor System

For the sake of standardization and spare parts inventory all conveyor drives are equipped with equally powered squirrel-cage-induction motors, one motor for the boom conveyor, one motor for the bridge conveyor and three motors for the dump conveyor. Oil hydraulic couplings are incorporated as a starting aid for the boom conveyor and the bridge conveyor. The conveyors are reversible in order to counteract chute blockage. 3 water-cooled hydraulic couplings are provided for the 3 drive units of the dump conveyor. These serve as a starting aid and for infinitely variable control of the transport speed from 3.7 m/s to 5.3 m/s. Service cars are provided for removal and re-installation of the upper and lower conveyor idlers. The boom of the connecting bridge and the main girder of the belt tripper are designed such that the return out of the rubber conveyor belt is located underneath the support structure, thereby enabling dribbling and dirt to fall down freely. Currently operated 3.2 m wide belt conveyor system with a speed of 7.5 m/s (maximum speed 15 m/s) have been tailored to the maximum transport volume occurring in today's opencast mining operations.

Crawler Track Assembly

The crawler mechanisms are driven by the slip-ring induction motors and planetary gearbox. The crawlers of the belt tripper are mechanically linked in pairs, a squirrel cage steering motor driving the guarded steering spindles via propeller shafting and reduction gears. The steering drives are properly dimensioned to enable the crawlers to be turned in completely while the actual stacker is at a standstill. The stacker is equipped with two sets of steering crawlers; the third set of crawlers is stationary. Removable guards on the crawler beams protect the drive unit, the rollers, the take up system and the lubricating pipes from dirt.

Auxiliary Equipment

From four lubrication centre all lubricating points are automatically supplied with grease oil via centralized lubrication system. Lifting beams, cranes and electric lifting tackle with capacities 0.5t to 10 t are provided for maintenance and as handling aids. There is also a splicing unit for repair of rubber belts. The stacker system is equipped with a central fire fighting system which is connected by pipes to the central water supply system. Two standby tanks of 50001 capacities each are available for emergencies. Additional manual fire extinguishers are arranged at all important points e.g. at the drive units and in the electrical equipment houses.

For the operating personnel there are washroom with toilets, a crew room with kitchen, air condition operators' cabins and a workshop is provided for minor repair. A generalized schematic diagram of a stacker containing double boom (non-revolving balanced type) is shown in the

Fig. 9.2. Technical Specification of a Stacker is shown in the table -9.1. Table 9.2 highlights the specification of stackers manufactured by Voest- Alpine , Austria

Table 9.1 Technical Specification of a Stacker

Overall length of a stacker system	168 m
Overall height	35 m
Overall total install capacity of power	3800 kW
Outreach of discharge boom	50 m
Belt width of discharge conveyor	2.6m
No. of crawlers under stacker	2x3=6
No. of crawlers under belt conveyor	2x4=8
Maximum soil pressure of stacker	0.9 kg/cm^2
Maximum soil pressure of belt tripper	1.2 kg/cm^2
Width of crawler pads	3.8 m
Supporting width of connecting bridge	55.0 m + 2.5 m
Transport speed of conveyor system	3.5 m/s to 4.5 m/s
Travel speed of stacker system	3 m/min to 10 m/min
Slew speed of boom	30 m/min

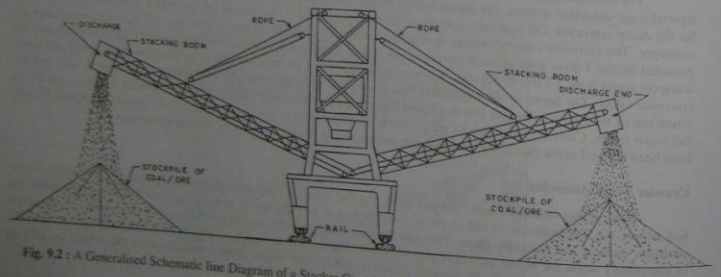


Fig. 9.2 : A Generalised Schematic line Diagram of a Stacker Containing Double Boom (Non Revolving Balanced Type.)

Table -9.2 Examples of other Stackers Manufactured by Voest- Alpine , Austria

Stackers	Model	Theoretical Production capacity (m^3/h)	Length of feed belt (m)	Length of discharge belt (m)	Belt width (mm)	Max. Discharge Length (m)	Installed Power (kW)	Service Weight (t)
ARS $\frac{1600}{20+30}$ x12.5	Rail mounted	5500	20	30	1600	12.5	900	308
ARS $\frac{1800}{30+45}$ x18	Crawler track mounted	5500	30	45	1800	18.0	1250	600
ARS $\frac{1600}{34+45}$ x15	Crawler track mounted	4800	34	45	1600	15.0	873	405
AR $\frac{1000}{30}$ x9.5	Crawler track mounted	1100	-	-	1000	9.5	165	32
AR $\frac{1600}{30}$ x10	Crawler track mounted	5500	-	-	1600	10.0	550	98

Example of a stacker Manufactured by Mannesnann Demag ,Germany :

Crawler track mounted stacker having a capacity of 2500m^3/h with a discharge height 14m and belt width 1400mm. Belt has a speed around 3m/s and length of discharge boom is 31m.

RECLAIMER

For reclaiming coal/ore/mineral from the stockpile various types of reclaimers are developed in the world, out of which the most common types are as following-

- Bucket wheel reclaimer
- Drum type reclaimer
- Scraper reclaimer (bridge and portal types)
- Portal girder reclaimer , etc.

Bucket Wheel Reclaimer (Fig. 9.3)

It consists of a rotating wheel having a number of buckets arranged around it's periphery. As the wheel rotates and dig into the stockpile the buckets scoop materials and discharge onto a belt conveyor for onward transmission to the bunker via. a series of belt conveyor system. The bucket wheel may either be mounted at the end of the reclaimer boom (this machine can swing horizontally) or may be suspended from a bridge, spanning the width of the stockpile. The rail mounted boom type reclaimer (Fig. 9.3) runs over a belt conveyor which runs parallel to the stockpile. The reclaimer after reclaiming the materials discharge them on to the belt conveyor run through the frame of its base. Crawler track mounted bucket wheel reclaimers are also available in the market. In rail mounted reclaimers hydraulic equalization by means of robust hydraulic rams are provided in the under carriage unit to stabilize the machine when there is the presence of differences of level between the two rails. It also provides necessary articulation to the machine if required. The slewing speed of the reclaimer boom can be varied with the help of thyristor controlled D.C. motors. For optimum digging operation the bucket wheel has to be arranged in designed stage at an angle of not more than 25^0 to the wheel boom conveyor belt. Superstructure can slew at an angle of ±360^0 and at a speed varying in between 0.1m/s to 0.5 m/s. The diameter of the bucket varies depending on the design parameters.. However diameter ranging from 0.8 m to more than 14 m having a number of buckets around 8 and capacity of each bucket ranging from 0.5 m^3 to 3.8 m^3 are very common. RPM of the bucket is generally in between 5 to 7. An output of around 1000 m^3 to 10000 m^3 per hour by a

Safety in Spreaders and Conveyor Bridges

- Structural members of conveyor bridges, overburden spreaders and excavating machines, together with all ladders and platforms thereon should be cleaned off before the commencement of every shift.
- All automatic control and remote-control systems should be provided with interlocking devices to interrupt the power supply in case of defective operation.
- Conveyor bridges and overburden spreaders should be equipped with instruments for automatic continuous measurements of wind velocity and direction, interlocked with an emergency signaling system and a system of undercarriage control of overburden spreaders, as well as control and measuring instruments, limit switches, signaling and intercommunication devices. In addition to automatic brakes, the bridge undercarriages must be equipped with intact hand brakes. No machine should be operated unless the above-mentioned instruments, switches and devices are in good working order.
- During repairs to a conveyor bridge, the simultaneous disassembly of the automatic brakes and the hand brakes should be prohibited.
- Any counterweight located close to a road or traveling way should be effectively guarded.
- Conveyor lines on conveyor bridges and overburden spreaders should be provided with servicing platforms guarded on both sides.
- No passageway along a conveyor should be less than 700 mm in width.
- During inclement weather, thunderstorms, snowstorms, heavy rain or fog, when the visibility will be less than 25 m, the passage of persons or work on a conveyor bridge should be suspended.
- No conveyor bridge should be permitted to approach a structure or any mining and transport equipment to within a distance of less than 1 m or to operate in a position above other operating mining and transport equipment.
- Conveyor bridge railway tracks should not be used when they are under water.
- When an overburden spreader of the walking or railway-walking type is in motion, no persons, nor transport vehicles, machines or other equipment should be permitted to pass under the dumping cantilever of a conveyor bridge.
- The vertical distance between the end of a dumping cantilever of a conveyor bridge and the crest of the dump should not be less than 3 m. For overburden spreaders which move periodically and use a cantilever-type belt conveyor, this distance should not be less than 1.5 m.
- If there are signs of a rock slide on a waste dump, a conveyor bridge should immediately be removed from the danger zone.

Table 9.3 Spreaders being used in Neyveli Lignite Corporation Ltd.

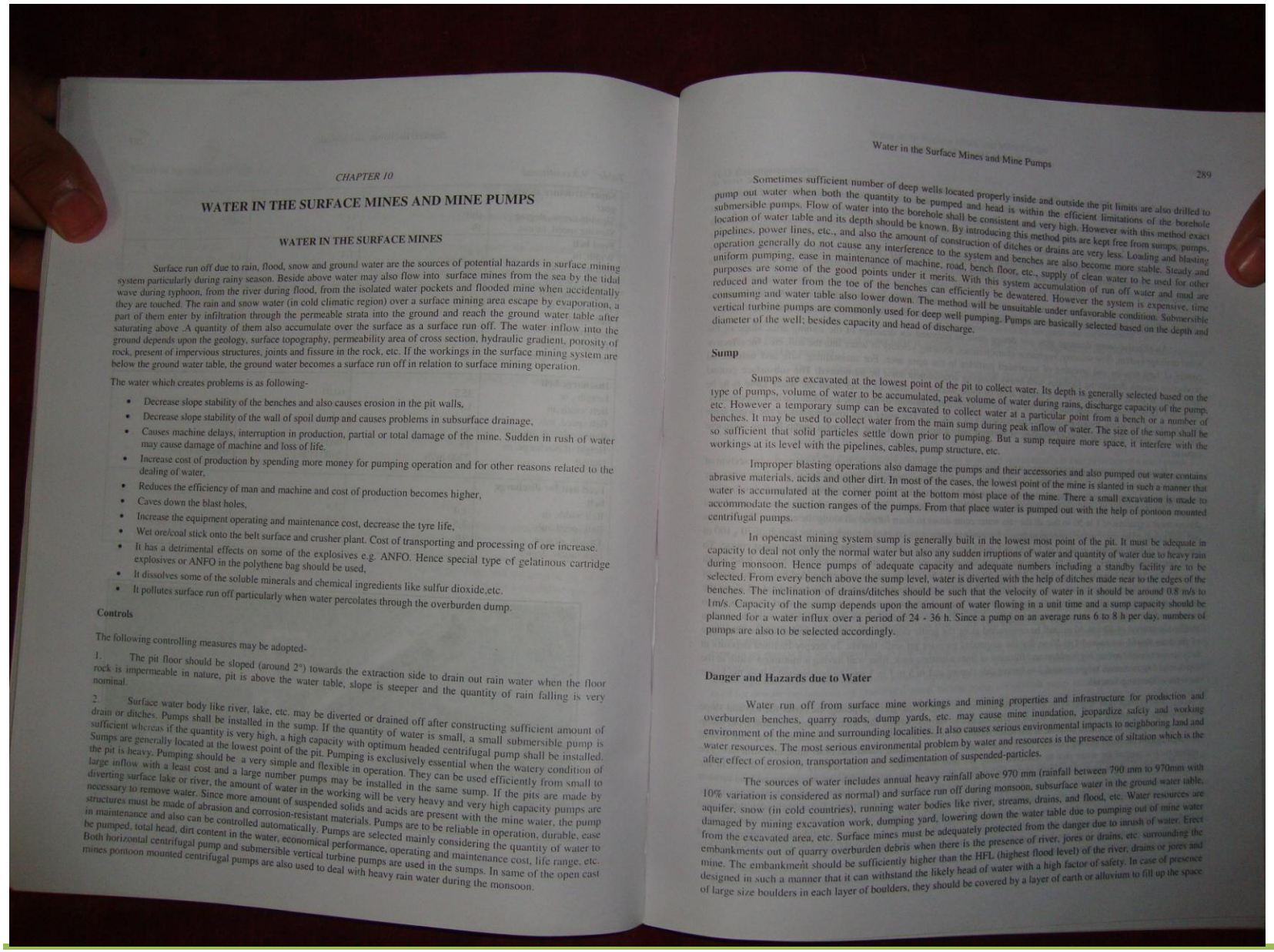
Type	I	II	III
Parameters	BAU 181 and 182	BAU 213	BAU 33
Travel gear	Two steerable crawler with one drive each, one rear crawler with drive	Two steerable crawlers with one non-steerable rear crawler with drive	Two single track crawler with transverse pivoted arm extended 3 point support
Width of crawler plates, m	1.5	2.5	3.0
Travel speed, m/min	8	8	6
Minimum radius for curve travel, m	50	50	-
Slewing gear of central part			
Slewing range, degree	360^0	360^0	360^0
Slewing speed, m/min	4.3	5	5
Diameter of ball race, m	3.72	5.88	-

Table : 9.3 continued

Super structure slewing gear			
Slewing range , degree	100^0	115^0	210^0
Slewing speed, m/min	19	15	-
Feed belt			
Width, m	1.4	1.4	1.8
Belt speed ,m/s	3.5	3.5	4.5
Trough , degree	16^0	-	-
Connecting belt			
Outer reach, m	15.35	23.26	-
Belt width ,m	1.4	1.4	1.8
Belt speed, m/s	3.8	3.8	-
Trough , degree	30	-	40
Inclination ,degree	18	18	18
Discharge belt			
Length , m	35.5	60.05	-
Belt width ,m	1.4	1.4	1.8
Belt speed, m/s	6	6.5	6
Belt trough , degree	30	-	40
Height of discharge, m	15	60(at 18^0 inclination)	40
Output	2700 m^3/h	2700 m^3/h	6000/8000 tph
Feed belt for discharge belt			
Belt width, m	1.4	1.4	1.8
Belt speed ,m/s	3.5	3.8	-
Trough ,degree	16	-	40
Inclination ,degree	4	-	-

A Spreader under Operation

By:- Vikas shrimali & Vipul bansal



CHAPTER 10

WATER IN THE SURFACE MINES AND MINE PUMPS

WATER IN THE SURFACE MINES

Surface run off due to rain, flood, snow and ground water are the sources of potential hazards in surface mining system particularly during rainy season. Beside above water may also flow into surface mines from the sea by the tidal wave during typhoon, from the river during flood, from the isolated water pockets and flooded mine when accidentally they are touched. The rain and snow water (in cold climatic region) over a surface mining area escape by evaporation, a part of them enter by infiltration through the permeable strata into the ground and reach the ground water table after saturating above .A quantity of them also accumulate over the surface as a surface run off. The water inflow into the ground depends upon the geology, surface topography, permeability area of cross section, hydraulic gradient, porosity of rock, present of impervious structures, joints and fissure in the rock, etc. If the workings in the surface mining system are below the ground water table, the ground water becomes a surface run off in relation to surface mining operation.

The water which creates problems is as following-

- Decrease slope stability of the benches and also causes erosion in the pit walls,
- Decrease slope stability of the wall of spoil dump and causes problems in subsurface drainage,
- Causes machine delays, interruption in production, partial or total damage of the mine. Sudden in rush of water may cause damage of machine and loss of life.
- Increase cost of production by spending more money for pumping operation and for other reasons related to the dealing of water,
- Reduces the efficiency of man and machine and cost of production becomes higher,
- Caves down the blast holes,
- Increase the equipment operating and maintenance cost, decrease the tyre life,
- Wet ore/coal stick onto the belt surface and crusher plant. Cost of transporting and processing of ore increase.
- It has a detrimental effects on some of the explosives e.g. ANFO. Hence special type of gelatinous cartridge explosives or ANFO in the polythene bag should be used,
- It dissolves some of the soluble minerals and chemical ingredients like sulfur dioxide,etc.
- It pollutes surface run off particularly when water percolates through the overburden dump.

Controls

The following controlling measures may be adopted-

1. The pit floor should be sloped (around 2°) towards the extraction side to drain out rain water when the floor rock is impermeable in nature, pit is above the water table, slope is steeper and the quantity of rain falling is very nominal.

2. Surface water body like river, lake, etc. may be diverted or drained off after constructing sufficient amount of drain or ditches. Pumps shall be installed in the sump. If the quantity of water is small, a small submersible pump is sufficient whereas if the quantity is very high, a high capacity with optimum headed centrifugal pump shall be installed. Sumps are generally located at the lowest point of the pit. Pumping is exclusively essential when the watery condition of the pit is heavy. Pumping should be a very simple and flexible in operation. They can be used efficiently from small to large inflow with a least cost and a large number pumps may be installed in the same sump. If the pits are made by diverting surface lake or river, the amount of water in the working will be very heavy and very high capacity pumps are necessary to remove water. Since more amount of suspended solids and acids are present with the mine water, the pump structures must be made of abrasion and corrosion-resistant materials. Pumps are to be reliable in operation, durable, ease in maintenance and also can be controlled automatically. Pumps are selected mainly considering the quantity of water to be pumped, total head, dirt content in the water, economical performance, operating and maintenance cost, life range, etc. Both horizontal centrifugal pump and submersible vertical turbine pumps are used in the sumps. In same of the open cast mines pontoon mounted centrifugal pumps are also used to deal with heavy rain water during the monsoon.

Sometimes sufficient number of deep wells located properly inside and outside the pit limits are also drilled to pump out water when both the quantity to be pumped and head is within the efficient limitations of the borehole submersible pumps. Flow of water into the borehole shall be consistent and very high. However with this method exact location of water table and its depth should be known. By introducing this method pits are kept free from sumps, pumps, pipelines, power lines, etc., and also the amount of construction of ditches or drains are very less. Loading and blasting operation generally do not cause any interference to the system and benches are also become more stable. Steady and uniform pumping, ease in maintenance of machine, road, bench floor, etc., supply of clean water to be used for other purposes are some of the good points under it merits. With this system accumulation of run off water and mud are reduced and water from the toe of the benches can efficiently be dewatered. However the system is expensive, time consuming and water table also lower down. The method will be unsuitable under unfavorable condition. Submersible vertical turbine pumps are commonly used for deep well pumping. Pumps are basically selected based on the depth and diameter of the well; besides capacity and head of discharge.

Sump

Sumps are excavated at the lowest point of the pit to collect water. Its depth is generally selected based on the type of pumps, volume of water to be accumulated, peak volume of water during rains, discharge capacity of the pump, etc. However a temporary sump can be excavated to collect water at a particular point from a bench or a number of benches. It may be used to collect water from the main sump during peak inflow of water. The size of the sump shall be so sufficient that solid particles settle down prior to pumping. But a sump require more space, it interfere with the workings at its level with the pipelines, cables, pump structure, etc.

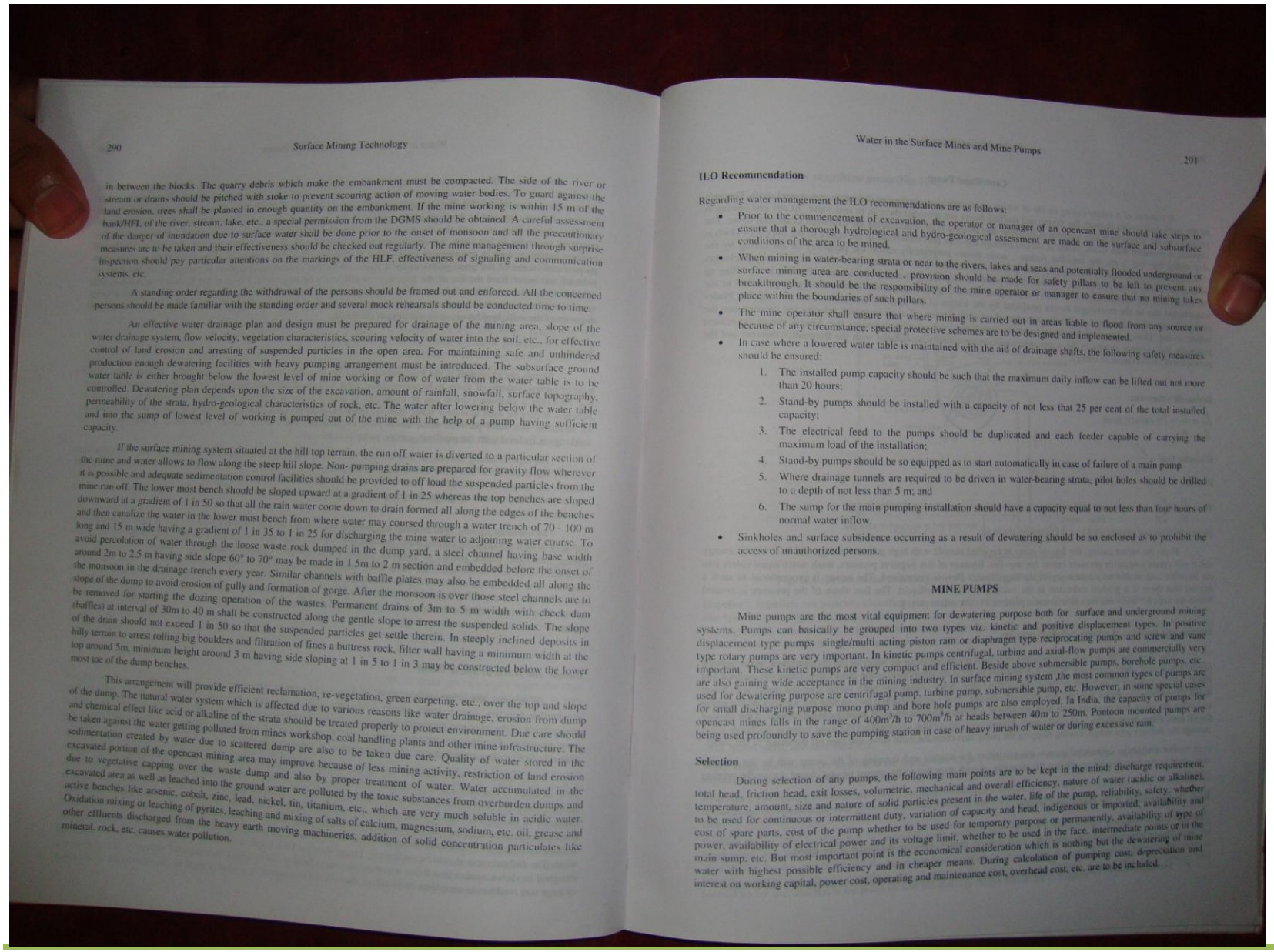
Improper blasting operations also damage the pumps and their accessories and also pumped out water contains abrasive materials, acids and other dirt. In most of the cases, the lowest point of the mine is slanted in such a manner that water is accumulated at the corner point at the bottom most place of the mine. There a small excavation is made to accommodate the suction ranges of the pumps. From that place water is pumped out with the help of pontoon mounted centrifugal pumps.

In opencast mining system sump is generally built in the lowest most point of the pit. It must be adequate in capacity to deal not only the normal water but also any sudden irruptions of water and quantity of water due to heavy rain during monsoon. Hence pumps of adequate capacity and adequate numbers including a standby facility are to be selected. From every bench above the sump level, water is diverted with the help of ditches made near to the edges of the benches. The inclination of drains/ditches should be such that the velocity of water in it should be around 0.8 m/s to 1m/s. Capacity of the sump depends upon the amount of water flowing in a unit time and a sump capacity should be planned for a water influx over a period of 24 - 36 h. Since a pump on an average runs 6 to 8 h per day, numbers of pumps are also to be selected accordingly.

Danger and Hazards due to Water

Water run off from surface mine workings and mining properties and infrastructure for production and overburden benches, quarry roads, dump yards, etc. may cause mine inundation, jeopardize safety and working environment of the mine and surrounding localities. It also causes serious environmental impacts to neighboring land and water resources. The most serious environmental problem by water and resources is the presence of siltation which is the after effect of erosion, transportation and sedimentation of suspended-particles.

The sources of water includes annual heavy rainfall above 970 mm (rainfall between 790 mm to 970mm with 10% variation is considered as normal) and surface run off during monsoon, subsurface water in the ground water table, aquifer, snow (in cold countries), running water bodies like river, streams, drains, and flood, etc. Water resources are damaged by mining excavation work, dumping yard, lowering down the water table due to pumping out of mine water from the excavated area, etc. Surface mines must be adequately protected from the danger due to inrush of water. Erect embankments out of quarry overburden debris when there is the presence of river, jores or drains, etc. surrounding the mine. The embankment should be sufficiently higher than the HFL (highest flood level) of the river, drains or jores and designed in such a manner that it can withstand the likely head of water with a high factor of safety. In case of presence of large size boulders in each layer of boulders, they should be covered by a layer of earth or alluvium to fill up the space



in between the blocks. The quarry debris which make the embankment must be compacted. The side of the river or stream or drains should be pitched with stoke to prevent scouring action of moving water bodies. To guard against the land erosion, trees shall be planted in enough quantity on the embankment. If the mine working is within 15 m of the bank/HFL of the river, stream, lake, etc., a special permission from the DGMS should be obtained. A careful assessment of the danger of inundation due to surface water shall be done prior to the onset of monsoon and all the precautionary measures are to be taken and their effectiveness should be checked out regularly. The mine management through surprise inspection should pay particular attentions on the markings of the HLF, effectiveness of signaling and communication systems, etc.

A standing order regarding the withdrawal of the persons should be framed out and enforced. All the concerned persons should be made familiar with the standing order and several mock rehearsals should be conducted time to time.

An effective water drainage plan and design must be prepared for drainage of the mining area, slope of the water drainage system, flow velocity, vegetation characteristics, scouring velocity of water into the soil, etc., for effective control of land erosion and arresting of suspended particles in the open area. For maintaining safe and unhindered production enough dewatering facilities with heavy pumping arrangement must be introduced. The subsurface ground water table is either brought below the lowest level of mine working or flow of water from the water table is to be controlled. Dewatering plan depends upon the size of the excavation, amount of rainfall, snowfall, surface topography, permeability of the strata, hydro-geological characteristics of rock, etc. The water after lowering below the water table and into the sump of lowest level of working is pumped out of the mine with the help of a pump having sufficient capacity.

If the surface mining system situated at the hill top terrain, the run off water is diverted to a particular section of the mine and water allows to flow along the steep hill slope. Non- pumping drains are prepared for gravity flow wherever it is possible and adequate sedimentation control facilities should be provided to off load the suspended particles from the mine run off. The lower most bench should be sloped upward at a gradient of 1 in 25 whereas the top benches are sloped downward at a gradient of 1 in 50 so that all the rain water come down to drain formed all along the edges of the benches and then canalize the water in the lower most bench from where water may coursed through a water trench of 70 - 100 m long and 15 m wide having a gradient of 1 in 35 to 1 in 25 for discharging the mine water to adjoining water course. To avoid percolation of water through the loose waste rock dumped in the dump yard, a steel channel having base width around 2m to 2.5 m having side slope 60° to 70° may be made in 1.5m to 2 m section and embedded before the onset of the monsoon in the drainage trench every year. Similar channels with baffle plates may also be embedded all along the slope of the dump to avoid erosion of gully and formation of gorge. After the monsoon is over those steel channels are to be removed for starting the dozing operation of the wastes. Permanent drains of 3m to 5 m width with check dam (baffles) at interval of 30m to 40 m shall be constructed along the gentle slope to arrest the suspended solids. The slope of the drain should not exceed 1 in 50 so that the suspended particles get settle therein. In steeply inclined deposits in hilly terrain to arrest rolling big boulders and filtration of fines a buttress rock, filter wall having a minimum width at the top around 5m, minimum height around 3 m having side sloping at 1 in 5 to 1 in 3 may be constructed below the lower most toe of the dump benches.

This arrangement will provide efficient reclamation, re-vegetation, green carpeting, etc., over the top and slope of the dump. The natural water system which is affected due to various reasons like water drainage, erosion from dump and chemical effect like acid or alkaline of the strata should be treated properly to protect environment. Due care should be taken against the water getting polluted from mines workshop, coal handling plants and other mine infrastructure. The sedimentation created by water due to scattered dump are also to be taken due care. Quality of water stored in the excavated portion of the opencast mining area may improve because of less mining activity, restriction of land erosion due to vegetative capping over the waste dump and also by proper treatment of water. Water accumulated in the excavated area as well as leached into the ground water are polluted by the toxic substances from overburden dumps and active benches like arsenic, cobalt, zinc, lead, nickel, tin, titanium, etc., which are very much soluble in acidic water. Oxidation mixing or leaching of pyrites, leaching and mixing of salts of calcium, magnesium, sodium, etc. oil, grease and other effluents discharged from the heavy earth moving machineries, addition of solid concentration particulates like mineral, rock, etc. causes water pollution.

ILO Recommendation

Regarding water management the ILO recommendations are as follows:

- Prior to the commencement of excavation, the operator or manager of an opencast mine should take steps to ensure that a thorough hydrological and hydro-geological assessment are made on the surface and subsurface conditions of the area to be mined.
- When mining in water-bearing strata or near to the rivers, lakes and seas and potentially flooded underground or surface mining area are conducted , provision should be made for safety pillars to be left to prevent any breakthrough. It should be the responsibility of the mine operator or manager to ensure that no mining takes place within the boundaries of such pillars.
- The mine operator shall ensure that where mining is carried out in areas liable to flood from any source or because of any circumstance, special protective schemes are to be designed and implemented.
- In case where a lowered water table is maintained with the aid of drainage shafts, the following safety measures should be ensured:
    1. The installed pump capacity should be such that the maximum daily inflow can be lifted out not more than 20 hours;
    2. Stand-by pumps should be installed with a capacity of not less that 25 per cent of the total installed capacity;
    3. The electrical feed to the pumps should be duplicated and each feeder capable of carrying the maximum load of the installation;
    4. Stand-by pumps should be so equipped as to start automatically in case of failure of a main pump
    5. Where drainage tunnels are required to be driven in water-bearing strata, pilot holes should be drilled to a depth of not less than 5 m; and
    6. The sump for the main pumping installation should have a capacity equal to not less than four hours of normal water inflow.
- Sinkholes and surface subsidence occurring as a result of dewatering should be so enclosed as to prohibit the access of unauthorized persons.

MINE PUMPS

Mine pumps are the most vital equipment for dewatering purpose both for surface and underground mining systems. Pumps can basically be grouped into two types viz. kinetic and positive displacement types. In positive displacement type pumps single/multi acting piston ram or diaphragm type reciprocating pumps and screw and vane type rotary pumps are very important. In kinetic pumps centrifugal, turbine and axial-flow pumps are commercially very important. These kinetic pumps are very compact and efficient. Beside above submersible pumps, borehole pumps, etc., are also gaining wide acceptance in the mining industry. In surface mining system ,the most common types of pumps are used for dewatering purpose are centrifugal pump, turbine pump, submersible pump, etc. However, in some special cases for small discharging purpose mono pump and bore hole pumps are also employed. In India, the capacity of pumps for opencast mines falls in the range of 400m^3/h to 700m^3/h at heads between 40m to 250m. Pontoon mounted pumps are being used profoundly to save the pumping station in case of heavy inrush of water or during excessive rain.

Selection

During selection of any pumps, the following main points are to be kept in the mind: discharge requirement, total head, friction head, exit losses, volumetric, mechanical and overall efficiency, nature of water (acidic or alkaline), temperature, amount, size and nature of solid particles present in the water, life of the pump, reliability, safety, whether to be used for continuous or intermittent duty, variation of capacity and head, indigenous or imported, availability and cost of spare parts, cost of the pump whether to be used for temporary purpose or permanently, availability of type of power, availability of electrical power and its voltage limit, whether to be used in the face, intermediate points of in the main sump, etc. But most important point is the economical consideration which is nothing but the dewatering of mine water with highest possible efficiency and in cheaper means. During calculation of pumping cost, depreciation and interest on working capital, power cost, operating and maintenance cost, overhead cost, etc. are to be included.

Centrifugal Pump

It consists of a volute casing inside of which an impeller is incorporated mounting over a steel shaft. The ends of the steel shaft passes through the casing and stuffing boxes are provided there in to seal water. At one end bearing is provided in the water- tight cap whereas at the outer end bearing is provided inside the extension of the end cover. The impeller is provided with vanes. Water is sucked through the inlet port connected in the centre of the impeller (which is also called the 'eye ') .As the impeller rotates , water passes tangentially round the periphery of the casing by the centrifugal force generated by the impeller vane and thrown into the volute chamber from which water finally discharges through the delivery main (Fig.10.1). A centrifugal pump needs priming to create vacuum in it. Priming is the filling up the casing and the suction pipe line with water/liquid. When impeller rotates the kinetic energy is generated in the water/liquid due to the centrifugal forces produced by the blades of the impeller and water from the impeller blades (curved backward as compared d with the direction of rotation) thrown off at an increased velocity and pressure into the volute casing. Here because of the larger cross sectional area, the velocity decreases (i.e. kinetic energy decreases) gradually and pressure energy in the water increases correspondingly accordance with the Bernoulli's theorem and the principles of conservation of energy.

Bernoulli's theorem

$\frac{p}{g} = \frac{p}{w}$ = Pressure head

If kinetic head $= \frac{v^2}{2g}$

Potential head $= z$

Then $\frac{p}{w} + \frac{v^2}{2g} + z$ = constant , Where p=pressure in kg/cm^2 , v= velocity of water in m/s

g= acceleration due gravity =9.81 , z = Datum height in m

From the volute casing, the liquid/ water is expelled outside with high pressure energy and little kinetic energy and it will create a negative pressure inside the impeller. Because of this negative pressure, fresh water/liquid enters into the impeller and in this way continuity of the liquid/water flow is maintained. The casing is proportional in such a manner that there is a gradual reduction in the velocity of the water/liquid. The lion share of the pressure is created within the impeller itself and the rest amount is produced in the volute casing.

Starting and stopping of pumps including priming

After opening the air cock, if it is found that there is either little or no water in the suction range, water is to be poured into it until water overflows. Water will be filled up from the sump level to the main valve, Close the air cock and check for any leakage of air/water in the same suction range. After closing the main valve, start the pump with lower speed till the pressure gauge indicates the required pressure. This can be checked by opening the air cock of the pump and when the water will eject out with high pressure it will mean that the pump is working satisfactorily. After closing the air cock allow pump to run in the full speed. Always open (to combat overloading) the main valve to discharge water through the delivery pipe range. During stopping the pump first close the main valve and then off the motor of the pump.

For multistage centrifugal pumps the procedure for starting and stopping of the pump will be the same as mentioned in the forgoing paragraph but the air cocks of each stage has to be opened up and closed during the priming operation.

Basic characteristics of the centrifugal pumps

The basic characteristics of a centrifugal pump are as follows:

- The pressure which is developed by each impeller directly varies as the square of the speed.
- The quantity of discharge varies directly with the R.P.M. of the impeller.
- The requirement of power for a centrifugal pump varies directly as the product of the pressure and quantity.

Basic components of a centrifugal pump (Fig. 10.1)

A centrifugal pump consists of the following basic components-

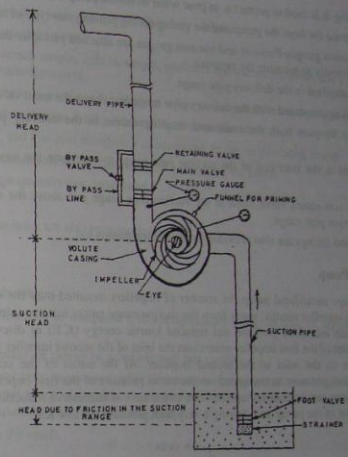


Fig. 10.1 : A Schematic Line Diagram of a Contrifugal Pump along with all Accessories.

Impeller : Impellers are basically classified into single suction impellers and double suction impellers. It is a circular disc type body having a series of backward curve vanes. Impellers are either closed type, open type or semi-open type. The impeller is mounted over a shaft and rotates inside the casing. The shaft is connected to the motor via gearbox and fluid coupling. Materials of impeller are bronze, cast steel, monel metal (to resist against the acid mine drainage), etc.

Casing : The casing covering the impeller is designed in such a manner that the kinetic energy of the water at the discharge end gradually converts into pressure energy and finally enters into delivery pipe range with high pressure energy and little kinetic energy. There are wide varieties of casings are available in the market out of which volute casing, vortex casing, casing having guide blades are very common. Materials which are used for manufacturing of casings are cast iron, cast steel, alloy steel, bronze (for corrosive water), etc.

Shaft : The shaft over which the impellers are mounted and which is connected to the drive unit is made from forged steel, monel metal, stainless steel, etc.

Wear rings : Provisions are also there for fitting wear rings on the impeller necks. This type of rings are made from monel metal, bronze, cast steel, stainless steel depending upon the condition of uses.

Bearings : The bearing housings are carried in rigidly constructed brackets, cast integrally with the pump casing and the housings are dowelled to provide positive location. Both ball and roller bearings are used in both the drive and non drive ends.

Stuffing box : The stuffing box is provided with conventional packed glands using lubricated cotton packing. Mechanical seals are also sometimes provided. The gland is of split construction which can easily be removed for accessibility for the packing removal. A lantern ring is fitted to receive the pumped liquid at delivery pressure to lubricate packing and also to provide a seal to prevent the ingress of air. Besides above some other fittings and accessories are also attached in the pump suction and delivery ranges which are as follows:

- Suction and delivery pipes-They are fitted in the suction and delivery end of the pump inlet and outlet respectively.
- Funnel for priming-It is used to prime i.e. to pour water to fill the pump and suction range.
- Air cocks-To release air from the pump and the casing during priming this type of air cocks are used.
- Pressure and vacuum gauges-Pressure and vacuum gauges are attached just after the pump outlet and before the pump inlet respectively to measure the pressure.
- Main valve-It is attached in the delivery pipe range.
- Retaining valve-It is connected with the delivery pipe range just above the main valve.
- By-pass valve-To by-pass both the main and retailing valves in the delivery pipe range by-pass valve is provided.
- Strainer-It is fitted at the inlet end of the suction pipe range to segregate out any solid particles mixed with water.
- Foot valve-It is a non-return valve fitted in the suction pipe range just above the strainer to retain water/fluid column in the suction pipe range.

All the above accessories and fittings are also provided in the turbine pumps too.

Multi-Stage-Centrifugal Pump

In case of multistage centrifugal pump the number of impellers mounted over the shaft is more than one (Fig. 10.2). So in this case when impeller rotates, water from the suction range enters into the inlet of the first impeller and is discharged at the outlet with increased pressure and reduced kinetic energy (K.E) as discussed previously. The high pressure water from the outlet of the first impeller enters into the inlet of the second impeller through the pipe connecting outlet of the first impeller to the inlet of the second impeller. At the outlet of the second impeller there is the development of further higher pressure as compared to the outlet pressure of the first impeller. In this way pressure of water goes on increasing from one impeller to another one at the cost of successive reduction of K.E. Finally the water liquid comes out of the outlet of the pump with having a very high pressure but low K.E.

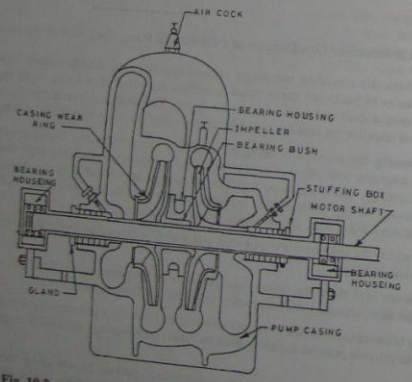


Fig. 10.2 : A Schematic Line Diagram of a Typical Centrifugal Pump.

Advantages and disadvantages of a centrifugal pump

A centrifugal pump offers the following main advantages-

- It produces uniform pressure with wide range of capacity variation.
- Higher velocity in the suction range.
- The pump is very compact, simple, light in weight, produces less vibration, noise, and also the pump has only fewer mechanical parts/components.
- Initial capital requirement to purchase the machine is less as compared to a reciprocating pump.
- It does not produce any pulsation during discharging of water/liquid.
- It requires less space, and low cost foundation as compared to the reciprocating pump.
- Load on the motor is more uniform as compared to a reciprocating pump.
- It is applicable for high quantity and small head.
- Installation is easy.
- It's operating characteristics are also very much flexible.

Disadvantages

It has the following main disadvantages-

- It has the lower efficiency as as compared d to a reciprocating pump. In case of higher viscosity it's efficiency further decreases.
- Very high discharge pressure cannot be achieved with this system.
- Shorter life, high power cost, requirement of skilled manpower for operation and maintenance, inaccessibility in the internal parts, higher speed susceptible to greater wear in the pump parts, etc. are the other inherent demerits in this system as compared to the reciprocating pumps.
- Suction lift should be the small.

Uses

The centrifugal pumps are available in the range starting from very small quantity to as much as quantity 20000 m^3/hour and head ranging from 5m to 200 m. However, higher head is possible with multistage pumps and higher speed motors. Centrifugal pumps are used in mining industry for dewatering from the mine sump, water logged area, main dip during shaft sinking operation, etc. Beside above this type of pumps are used for coal/mineral washing, leaching, waste disposal, boiler feed, spraying purposes, feeding in the condenser of the mine air compressors, etc. The specification of some of the centrifugal mine pumps are given below (Table 10.1).

Table : 10.1 Specification of some of the mine pumps

Pumps	Quantity (l/s)	Head (m)	Power (kW)	Voltage (Volt.)	Material used	Price Rupees in lakhs (year 2003)
Main Pumps Centrifugal (Flame Proof)						
	38	90	55	550	Bronze	4.30
	38	150	90	550	Stainless	6.60
	38	200	110	550	Bronze	7.30
	60	300	265	3300	Bronze	19.00
	75	300	310	3300	Bronze	21.00
	80	120	135	3300	Stainless	14.50

Table 10.1 continued

Face Pumps Centrifugal (Flame Proof)	15	30	7.5	550	Special Alloy	1.00
	20	60	22	550	Special Alloy	1.10
Submersible Pumps	30	100	45	550	Bronze	2.80
	38	100	90	550	Bronze	5.80
	60	230	260	550	Bronze	26.60

POWEN pump factory of Poland manufactures various types of centrifugal pumps out of which HC series pumps are worth to mention. These pumps are heavy duty ,single stage horizontal pumps with closed impellor. These pumps are compact and robust design , able to pump mixtures containing up to 50% of solids by volume, impeller neck adjustable, hydrodynamic shaft seal , no external water needed , can operate in series and very easy to maintain. Table 10.2 shows some of the centrifugal pumps manufactured by the POWEN pump factory.

Table : 10.2

Pump type	Capacity (m^3/h)	Head (m)	Power (kW)	r.p.m	Maximum solid size (mm)
HC125	250	45	44	1470	30
HC150	400	63	100	1470	25
HC200	800	115	300	1470	44

Turbine Pump (Fig. 10.3)

A turbine pump consists of a number of impellers which are mounted over the same shaft. When impellers rotate, water from the suction range enters into the first impeller via the 'eye' of the impeller and is discharged at the outlet with increased pressure and less K.E. The outlet water from the first impeller enters into the inlet of the second impeller through the stationery diffusing channels of the diffuser surrounding the impeller. In the diffusing channel velocity of the water is reduced and the K.E is converted into the pressure energy, intensity of which depends upon the velocity of water ejected out from the impeller. The same process is repeated again and again during movement of water from one impeller to the other and ultimately water is delivered into the delivery pipe range with a very high pressure but with a little velocity.

This development of high pressure is the sum of the head developed at each stage (each impeller) i.e. in other word the total effect is cumulative. Maximum amount (more than 50%) of increase in pressure takes place in the impellers and the rest amount in the diffusing channels. Difference between the multistage centrifugal pump and the turbine pump is that, in the turbine pump the impellers are surrounded by a series of stationary guide diaphragms which are termed as diffusers. To counter balance the axial thrust the discs are balanced by reversing half the total number of impellers in the either half of the mounted shaft as shown in the figure (10.3). The main parts of a turbine pump is the pump casing, impellers, stuffing boxes, diffusing channels (these are formed by the diaphragm which are connected in the inner periphery of the outer casing), etc.

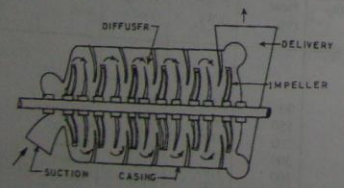


Fig. 10.3 : A Schematic Line Diagram of a Turbine Pump.

Water hammer

Water while flowing through a pipe posses some momentum on account of its motion. It has been experienced that if the flowing water is suddenly brought to rest by closing the valve, its momentum will be destroyed causing a very high pressure on the valve. This high pressure will be followed by a series of pressure vibrations. The pressure vibrations may set up noises in the pipeline which is known as knocking. Such a knocking often heard in water pipe if a tap is turned off quickly. The sudden rise of pressure in the pipeline is known as hammer blow or water hammer. Sometimes this hammer blow is so high that it may even burst out the pipe. It is thus obvious that the valves of the pipeline or penstocks should always be closed gradually.

At the time the valve is being closed, the fluid in the pipe is moving possessing kinetic energy (K.E). With sudden closure of valve the K.E has to be converted into internal energy with a rise in pressure. Actually a sudden pressure pulse is formed at the valve and this pulse travels backwards at the speed of sound wave setting up reflected pulses. This behavior of the flow is termed as the unsteady flow. The magnitude of the pressure rise and falls depends upon the speed at which the valve is closed. In the process of slow closing of valve the whole mass of fluid in the pipeline is decelerated and the inertial pressure depends upon the length of pipe and the time taken in closing the valve which results the deceleration of water. However, in a sudden closure of valve the initial pressure created is independent of length and time and depend upon the initial velocity of fluid and the velocity of sound wave in the fluid. The elastic properties of pipe material and the flowing fluid are also taken into account for deriving the equation of the inertial pressure.

Cavitations

If the pressure at any point inside a pump drops below the vapor pressure corresponding to the temperature of the liquid, the liquid will vaporize and from cavities of vapor. The vapor bubble are carried along with the steam until a region of higher pressure is reached when they collapse with a tremendous shock on the adjacent walls. This phenomenon is called the cavitations. The sudden inrush of liquid into the cavity created by the collapsed vapor bubbles causing mechanical destruction sometimes apparent as a boring action which might be called erosion. Chemical reaction between the gases and metal also occurs and results in corrosion and additional destruction of the metal. Other undesirable feature is the accompanying noise, varying in different units from low rumbling to loud knocks which result heavy vibration of the unit. The required acceleration of water to the high velocity is suddenly filling the hollow spaces is a loss and cavitations will therefore be accompanied by a reduced efficiency.

Cavitations will occur mainly at the vane inlet portion of the pump impeller on both the vanes and the sidewalls. The erosion and wear due to cavitations will not occur at the point of lowest pressure where the gas pockets are formed but they occur at the point of further upstream.

To prevent cavitations, the suction pressure must be kept above the critical pressure at which it starts. Beside this ,the followings are some of the measures against the formation of cavitations.

- Velocity of fluid in the suction line should be low.
- Avoid sharp bends to keep the friction and turbulence losses at a minimum.
- Keep the temperature of the fluid at the minimum level.
- Specific speed of fluid in the inlet side should be kept at low level.
- If the thickness of inlet vane is small, smooth and the inlet edges are in aerofoil shape, cavitations formation will be low.
- There should be enough amount of vanes in the impeller for efficient guidance of water/liquid and also for lowering down the pressure of the fluid.
- Noise and erosion may be reduced if a small but optimum quantity of air is inserted into the suction side.

Question-1

A turbine pump discharges 2100 *l*/min with a total head of 200 m. If pump efficiency is 70%, motor efficiency is 90%. Calculate the power required. (Second Class (restricted), 1984)

Solution

2100 l/min = 35 l/s

$$\text{Power requirement} = \frac{Discharge(\frac{l}{s}) \times Head(m)}{102 \times efficiency\ of\ pump \times motor\ efficiency} = \frac{35 \times 200}{102 \times 0.9 \times 0.7} = 108.93 kW$$

Question-2

Calculate the electrical power required to lift 2000 l of water per minute to a height of 240 m. Assume pump efficiency as 0.75 and motor efficiency as 0.90. (Second Class (restricted), 1986)

Solution

2000 l/minute =33.33 l/s

$$\text{Power requirement} = \frac{33.33 \times 240}{102 \times 0.75 \times 0.9} = 116.18\ kW$$

Reciprocating Pump (Fig. 10.4)

In this type of pump a piston moves to and fro by a prime mover inside a cylinder which consists of an inlet and outlet port attached with the valve mechanism. In single stage reciprocating pump by each revolution of the prime mover the piston sucks water/liquid through the inlet valve by the backward stroke, compresses it from initial to the final pressure by the forward stroke and finally delivers it through the outlet valve in a single cylinder only. During the backward stroke a vacuum will be created inside the cylinders which will initiates to force water to fill the vacuum by opening the inlet valve. A reciprocating pump basically consist of a piston, cylinder, suction and delivery valves, suction pipe and delivery pipe, etc.

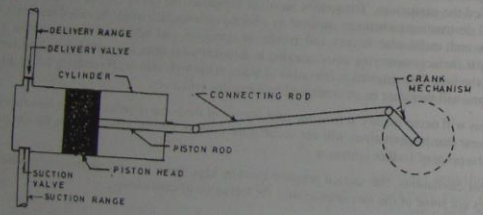


Fig. 10.4 : A Schematic Line Diagram of a Reciprocating Pump.

Triple Plunger Pump

In consist of 3 single acting plunger pumps driven by crosshead pistons which are operated through the connecting rods by the crankshaft. The connecting rods are staggered by 120° on the crankshaft. Each pump consists of a plunger, a plunger case, suction and delivery valves, gland, stuffing box, etc. Water fluid passes through the suction channel to the suction valve and thereafter fills the high pressure plunger case. The pressurized fluid produced by the compression strokes of the plunger. Water fluid there after is fed through the pressure valves via the pressure channel into the pressure control device located at the upstream of the delivery line or directly fed into the delivery line. Reduction gear and crankshaft have a common oil chamber and the pump has pressure lubrication. The attachments and accessories which are attached with a reciprocating pump consists of a foot valve attached at the front end of the suction pipe, main valve at the delivery pipe range, retaining valve above the main valve and also the bypass valve for side tracking main and retaining valve, air cock, air vessel, etc.

Advantages and disadvantages of a reciprocating pump

Advantages

A reciprocating pump offers the following main advantages :

- It is very much suitable to develop a very high pressure.
- It has larger life, uniform higher efficiency, simple, lower power cost, more reliable, lower operating cost as compared to a centrifugal pump.
- Lower speed causes less wear in the components of the pump and also can be applied under adverse and variable pressure condition.
- It does not require any priming
- All the internal parts are very much accessible and also less trained manpower is necessary in this system.
- It can be used to handle muddy water for high head (a head of 700m is very common) and low capacity. It is efficiently used in the power pack systems.

Disadvantages

It has many disadvantages which are as follows

- It produces pulsating output. Motors/prime movers withstand more pulsations and shock loads.
- Initial purchasing cost, maintenance cost, etc. are higher as compared to a centrifugal pump.
- Velocity in the suction range depends upon the speed of the pump.
- The pump is bulky, heavier in weight, produces heavy vibration, noise and very complicated in construction with heavy amount of parts and components.
- It requires more space, requires robust and costly foundation to install and also installation is not very easy
- Its operating characteristics are not very much flexible in nature.

Starting and stopping of the pump

After opening the main valve start pump and during stopping first switch off motor and thereafter close the main valve. After a longer stoppage if pump is to be restarted a little bit priming will yield better result.

Submersible Pumps (Fig. 10.5)

The pump is a machine which can suck water up from a certain level and then deliver it under pressure where it is required to be used. This was accepted despite the fact that the height from which it can suck water is strictly limited by fundamental principles of hydraulics irrespective of the power of the machine. Latter a novel form of pump appeared in the market and this was designed to run under submerged conditions in the water eliminating the need for suction entyrely thus greatly enhances the efficiency of the operation. Unfortunately this means that a long vertical shaft was required running down the well from the motor situated above ground. The motor had to be carefully mounted and housed and the shaft is carefully lined up and carried in bearings in the well which is quite an expensive set up. The submersible motor idea revolutionary. It consists of a pump close coupled to a motor, the whole assembly being capable of submerge into water to almost at any depth.

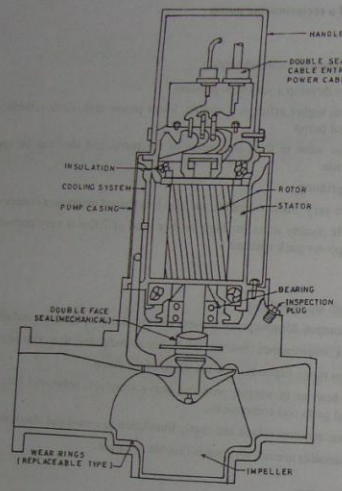


Fig. 10.5 : A Schematic Line Diagram of a Submersible Pump.

Brief Description

A brief description of the basic design is given below:

Pumps

The submersible pump basically comprises of a number of impellers mounted in a shaft, supported by journal bearings and the impellers are housed in suitable casings fitted with guide and return vanes. The pumped liquid/water enters into the first or lower most impeller and is passed progressively through the unit with an increase of head at each impeller stage and finally discharges into the raising main pipe work.

Submersible motors

Submersible motors are of the induction type, falling into two categories those using windings which are directly submerged in water generally referred to as wet motors and those which use the conventional dry motor winding being hermitically sealed from the surrounding liquid.

Dry motor

Here conventional dry windings are used and these are sealed within a suitable casing. The inner tube of which is non-magnetic one which separates the stator from the rotor unit and is only a few thousands of a centimeter thick. The stator of a dry machine is usually filled with resin to aid heat dissipation. Pumps are generally made of aluminum, cast iron, stainless steel, gunmetal, etc, and may be externally coated with chlorinated rubber point. The motor is totally enclosed, steel construction and in some version stator is designed to withstand. temperature up to 155^0 C. Three thermal protectors are connected in series in stator winding, one in each phase. These break the control circuit to the pump motor contactor if the temperature of the winding exceeds the stipulated temperature. Double mechanical face seals are provided with intermediate oil casing manufactured from carbon/tungsten carbide material. Motor has the PVC insulated

winding having PVC insulated power cable which is connected in the double sealed cable entry, although in some cases particularly in case of FLP submersible pump, power cables are factory mounted. There is a inspection cover on the top junction box for checking the electrical wiring.

Advantages

The main advantages of the submersible pumps are as follows :

- Submersible pumps are highly efficient for a small quantity of discharge,
- Does not require any housing for installation,
- Require very less space, very easy to install and withdraw,
- Very little supervision is required for maintenance,
- There is very less chance of explosion due to fire since pump is totally submerged into water,
- It does not require any priming,
- The impeller and diffuser are both adjustable for obtaining the optimum efficiency,
- It can be operated at any orientation and also automatic operation is generally practiced.

But the cable for a submersible pump is very expensive, it can be used efficiently to pump out clean water and also its working is not visible.

Common range of submersible pumps generally have power ranging from 0.9 kW to 425 kW having a discharge capacity varying from 25l/s to 5000 l/s with an average lift varying from 20m to 200m although a lift of around 300 m is not uncommon. Submersible pumps are also available to discharge a quantity of more than 7000 l/s. formed by the diaphragm which are connected in the inner periphery of the outer casing. Table 10.3 shows the characteristics of some of the submersible pumps.

Table : 10.3 Submersible pump characteristics

Head (m)	Allowable submergence (%)	Best range
38-54	40-65	60-65
>54-75	40-60	55-60
>75-110	37-55	50-55
195-230	35-45	40-45

POWEN pump factory of Poland manufactures many types of submersible pumps out of which P-Series is worth to mention. These pumps are centrifugal and single stage . With these pumps there is possibility of operation in submergence up to 6m deep, and also operation with self-sealing catch and tracks ,operation with liquid level sensors, etc. are possible Thermal protection of motor winding is incorporated in these type of pumps. Table 10.4 shows the specification of various pumps under this series.

Table -10.4 Some submersible pumps manufactured by the POWEN pump factory of Poland

Type	Capacity (m^3/h)	Head (m)	Max. solid size (mm)	r.p.m	Power kW
P30-F65/4A	50	13	28	1410	3.7
P55-K100/4A	100	13.5	75	1420	8.3
P110-Z125/4A	150	14.5	65	1455	11.3

Roto Pump (Fig. 10.6)

A roto pump (mono pump) basically operated on screwing action. .It is basically a single eccentric screw pump. It consist of a single helical monel /nitralloy/ Ni-Cr /stainless steel eccentric rotor unit placed inside the stationary torsion free bonded rubber/hypalon /other plastic material made stator. The rubber stator is placed inside a C.I/M.S. housing.

Pitch of the stator is half the pitch of a rotor unit and hence when rotor rotates inside the stator unit a constant seal starts advancing along with the entrapped water in it's front (from suction to the delivery side) uniformly. During rotating of the rotor, because of the exhaust created by the rotor and stator mechanism in the space in between them, all air exhaust out and water is filled up and caught in it's place by high suction. The trapped water thereafter moves forward as the rotor rotates and forced out through the outlet of the roto pump. The pumps are non-clogging and self priming type. Roto pumps are electrically driven, generally used for small capacity pumping varying from 0.5 m^3/hour to 120 m^3/hour at a pressure varying from 2.8 kg/cm^2 to 24 kg/cm^2. But a head of about 90 m is also not uncommon. Generally if the water is muddy and gritty , this type of pumps are very suitable. A snore by-pass is attached with the pump system which reticulates a very little quantity of water through the pump when there is no water in the sump and also it keeps the stator and rotor lubricated. The speed of the rotor is kept low (say in between 960rpm to 1450 rpm) and also the velocity of water/liquid inside the pump is very less. This helps lowering down the vibration, noise, wear in both the stator, rotor and other movable units. Roto pumps are very efficient as compared to centrifugal pump where high fluctuation of discharge of water is needed. By changing the direction of rotation the side of the suction and discharge can be reversed. However it should always be remembered that the diameter of delivery pipe range should be smaller as compared to the suction pipe range to increase the velocity of the water to combat suspension of the solid particles. If the pump is kept running in the dry condition then there will be more wear in the stator unit due to the lack of lubrication and the production of more heat. Sizes of the solid particles present in the water should be sufficiently smaller in size so that they can be accommodated in the helical spaces in between the stator and rotor. The driving shaft of rotor is hollow in construction which mounts over ball bearings and rotates by an electric motor via a fluid coupling and gearbox. The roto pump is of skid mounted. Table 10.5 shows specification of some of the mono pumps.

Table : 10.5 Specification of some of the mono pumps

Capacity (m^3/h)	Power (kW)	Head (m)	rpm	Type
10.60	1.65	50	600	LL-1
22.30	8.3	50	1200	
17.4	2.3	50	400	LM-1
36.8	5.5	50	800	

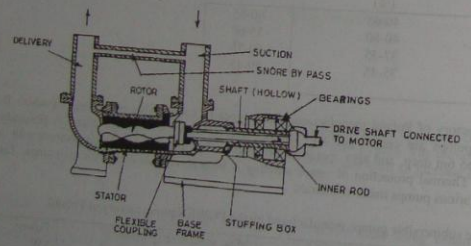


Fig. 10.6 : A Schematic Line Diagram of a Mono Pump (Roto Pump).

CHAPTER 11

CRUSHER, SCREEN, ETC.

CRUSHERS

Crushing is the process of reducing the size of the lump of ore or overburden rock into definite smaller sizes. Crushers crush the coarse lumps into finer crushing products by a group of operations but during crushing a substantial amount of fines are also produced. The crusher crushes the feed by some moving units against a stationery unit or against another moving unit by the applied pressure, impact, shearing or combine action of them. The strain in the feed material due to the sufficiently applied pressure, impact forces or shearing effect , causes fractures occur on them provided that the elastic limit of the feed material lower than the values as mentioned the parameters. The crushers are very much rugged, massive and heavy in design and contact surfaces are made of replaceable type high tensile manganese or other alloy steel sheet having either flat or corrugated surfaces. To guard against shock and overload, the crushers are provided with shearing pins or nest of heavy coiled springs. Crushers may be classified as following:

- Primary Crushers: These type of crushers has large opening in the input side and are used to crush the feed of run of mine (ROM) supplied directly from the mine. The output products are also relatively coarser in size. Example-Jaw crusher, gyratory crusher, etc, for hard rock formation and knobbed roll crusher for soft rock formation.
- Secondary Crushers: These type of crushers has comparatively smaller opening in the intake side and are used to crush the feed supplied from the primary crusher into more uniform and finer sizes. Example-cone crusher, reduction gyratory crusher, spring rolls, disk crushers, etc.
- Fine Crushers: These types of crushers are used to crush the supplied feed into more uniform and finer product. Example-gravity stamp.
- Crusher for Special Purposes: To reduce the production of much amount of fines and also for selective crushing of soft rocks these types of crushers may be used. Example-hammer mill.

Around 30-50% of total energy is used in an ore beneficiation plant. Crushing costs may be reduced by using large capacity crusher, improved control of crushing circuits and by using more reliable methods for the estimation of crusher performance. Wider use of mobile and semi-mobile crushers allow lower transport cost of crushed ore by conveyor belts instead of transporting entire ore by a truck to a fixed crushing section. Reduction in costs may also be effected by crushing to finer sizes in closed circuit with screens followed by single stage grinding.

JAW CRUSHER

Jaw crusher is a primary type of crusher which has two Jaws, out of which one is stationary and attached rigidly with the crusher frame whereas the other jaw moves by a small throw forward and retarded back successively to crush the ore or rock boulders (Fig- 11.1).

Basically the jaw crusher can be divided into two according to the amplitude of motion of the moving face. They are:

(a) Blake type jaw crusher-Here the movable jaw is hinged at the top of the crusher frame so that the maximum amplitude of motion is obtained at the bottom of the crushing jaw.

(b) Dodge type jaw crusher-Here the movable jaw is hinged at the bottom of the crusher frame so that the maximum amplitude of motion is obtained at the top of the crushing jaws.

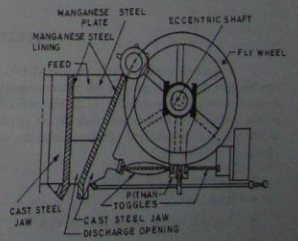


Fig. 11.1 : A Schematic Line Diagram of a Jaw Crusher.

For larger material crushing, always larger Blake type jaw crushers are selected. The characteristics of this type of crusher are as following:

- Larger, rough, blocky as well as sticky rock or ore lumps can be crushed,
- It has a very large opening,
- Reinforcement of the crusher is possible with the help of high strength crusher frame to crush very hard rock or ore lumps.
- It is very simple to adjust to prevent much of wear and also very easy to repair,
- Maintenance of the crusher is very easy.

Crushing of ore, mineral or rock depends upon the characteristics of ore, size of the feed, the discharge openings, speed, throw, nip angle (It is the angle-between the jaw faces. Generally it is around 20^0 to 23^0 in higher capacity jaw crusher), etc. of the crusher. The capacity of the crushing depends upon the reduction ratio (it is the ratio between the size of the feed and the size of the discharge. Higher the reduction ratio less will be the capacity of the crusher.) ,nip angle (increase in the angle will decrease the capacity of the crusher), increase in speeds, throw, curved shaped jaws, etc. The jaw crusher should not be buried by the feeding minerals or ores which will tend to chock the mouth of the crusher and often a power operated hook will be necessary to remove the ore or mineral lumps which jam the crusher unit. Generally average reduction ratio is around 1.8 to 7 with a maximum setting of gap is around 2 to 2.4. However this reduction ratio may vary depending upon many operating conditions. The jaws do not touch each other and have a wide gap at the top. The faces of the jaws are either flat or convex (convex jaws are better which reduces the frequencies of chocking and also increases the capacity of production). The movable jaws is fixed to a pitman or to an eccentric arm hinged on a spring fulcrum and toggle. To equalize the load one or two flywheels are mounted on the eccentric shaft mounted at the top of the pitman. As the shaft rotates either by flat or 'V' belt drive, the pitman and jaw moves towards the fixed jaw and then moves away from it successively. Rocks are pressed and crushed when the moving jaw is moved towards the fixed jaw and thereafter drops down as the jaw moves back. The crushing process is repeated till the ore or rock lumps are crushed to sufficient sizes to pass through the discharge opening. Various sizes of jaw crushers are available, even a crusher sizes of 1650 mm x 2150 mm (1650 mm is the width of the maximum opening at the top and the jaws are 2150 mm in long) are not uncommon. The maximum diameter of the feed is ranged in 80 % to 85% of the width of the maximum opening. Such a heavy crusher (1650mm x 2150mm) crushes rock, mineral or ore varying from 22.5cm to 30 cm with a capacity ranging from 420t to 630t per hour. The motor rpm and power are around 90 and 187.5 kW respectively. The jaw and the sides of the unit is lined with replaceable wear resistant plate liners. Basically the jaw crusher may be divided into two:

(a) Single toggle type :Here the number of toggle plate is only one. It is cheaper and has less weight as compared to a double toggle type jaw crusher. It is used for soften rock condition.

(b) Double toggle type: This type of jaw crusher has two toggles and used for heavy duty crushing.

GYRATORY CRUSHER

A gyratory crusher mainly consists of two vertical conical or domed shaped crushing member. The inner member(apex of which is pointing up) gyrates around the vertical axis inside the stationary outer crushing member(apex of which pointing downward) (Fig- 11.2). The inner conical member may gyrate equally at the top and also at the bottom or only gyrate at the bottom and crush the ore or mineral. Out of many verities available, the short shaft suspended-spindle and fixed-spindle type gyratory crusher are worth to mention. The main characteristics of this type of gyratory crusher are as following:

- It is a very simple machine.
- It has higher capacity
- Larger reduction ratio is possible by it.
- Reduction of higher speed is not necessary.
- Handling of ore or overburden rock throughout its periphery is possible.
- It is more economic.

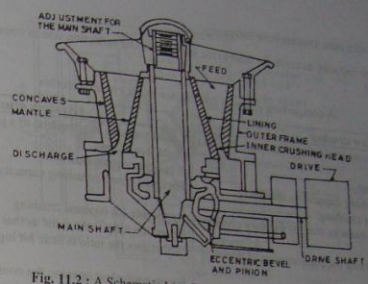


Fig. 11.2 : A Schematic Line Diagram of a Gyratory Crusher

The gyratory crusher can be used both as a primary or secondary crusher. As the feed materials pass downward direction through the gradual converging annular space between the fixed outer crushing member and the inner rotating member, they are crushed by the crushing surfaces when they narrow down the gap between each other and the crushed material flown out when both the crushing surfaces moves away from each other. The short shaft-suspended spindle gyratory crusher is a low height crusher, rugged in construction and due to shorter spindle height above the crusher easy maintenance and repair work are possible. This type of crusher is very good to crush soft and tough ore but not so efficient to crush hard rock. The speed of rotation of the inner member may vary between wide limits without increasing much of vibration and shock loads but it increases the rate of production. If the speed is increased, there will be the requirement of more amount of power. Increase in reduction ratio will be possible by decreasing the speed and the rate of feed. Higher speed decreases the tendency of clogging. The nip angle of the gyratory crusher varies approximately in between 21^0 to 22^0. Higher the nip angle, less will be the discharge capacity. Amount of throw in this type of crusher is less as compared to the jaw crusher and can be adjusted by changing the eccentric sleeves. Larger capacity crusher has higher throw rate. The capacity of the crusher depends upon the characteristics of ore, size of the feed discharge opening, speed, throw, nip angle, shape of the crushing surface, etc. Reduction ratio may be available varying from 1.5 to more than 7. Larger reduction ratio is available in larger crusher. Although the burying of crusher is not advocated but it will cause less bridging affect as compared to a jaw crusher. The speed of the inner cone member and the annular space in between the inner and outer cone should be properly selected so that falling of blocky or lumpy ore of rock to be crushed will be free and easy flowing. A gyratory crusher requires more power and require more vertical space for installation. It is also costlier as compared to a jaw crusher. But it produces more uniform products and also has higher production capacity. Various sizes of gyratory crushers are available, even a crusher size of 1800mm x 12100 mm (1800 mm is the width of maximum opening) having a capacity ranging in between 2500t to 3500 t per hour is not uncommon. The weight of this type of heavy crusher may be in between 625t to 700 t, with a power around 375 kW. The size of crushed output varies in between 22.5 cm to 30 cm with this type of heavy crusher.

CONE CRUSHER

It is a secondary gyratory type crusher having two members :

(a) A bowl shaped stationary member connected to the main frame,

(b) A flat conical or hemispherical shaped crushing head which is gyrated eccentrically to crush rock or ore lumps around a size range varying from 25- 35 cm to around 3 mm (Fig. 11.3). The outside stationary crushing surface flares out at the bottom instead of at the top as in the case of a gyratory crusher. This provides a larger discharge opening at the bottom for the crushed output. A nest of heavy coiled springs are so arranged in the crusher that in case of any metal enters along with the ore or rock particles into the crusher, it assists lifting of the bowl shaped outer crushing surface from the main frame to avoid any damages. Basically the cone crusher can be categorized into

(a) Standard-to crush coarser feeds
(b) Short Head-to crush finer feeds.

The main characteristics of a cone crusher is as following-

- It has a very high capacity to crush the feed to the same particular reduction size as compared to a roll crusher or gyratory crusher.

A comparison among a jaw crusher and a gyratory crusher

Jaw crusher	Gyratory crusher
1. Smaller capacity as compared to a gyratory crusher for the equal size of feed.	Larger capacity as compared to a jaw crusher for the equal size of feed.
2. It operates intermittently.	It operates continuously.
3. For equal power, crushing capacity is smaller.	For equal power, crushing capacity is higher.
4. Applicable for coarser crushing	Applicable for coarsest crushing
5. Reduction ratio is the same as that of gyratory crusher.	Reduction ratio is the same as that of jaw crusher (sometimes the ratio is little bit higher).
6. For an equivalent opening the consumption of power is more.	For an equivalent opening the consumption of power is less as compared to a jaw crusher.
7. Generally it is a primary crusher.	It can also be used as a secondary crusher e.g. reduction gyratory.
8. It is preferred while the average size of the lumps to be crushed are small.	It is preferred while the average size of the lumps to be crushed are larger enough.
9. Material can be fed from a particular side.	Material can be fed from all the sides.
10. Crushed product is not so uniform.	Crushed product is more uniform.
11. Vibration and shock loads are more. Higher speed causes more vibration and shock load.	Vibration and shock loads are less. Higher speed generally does not cause much vibration and shock load.
12. It requires less vertical space for installation.	It requires more vertical space for installation.
13. Cost of the crusher is less as compared to a gyratory crusher.	Cost of the crusher is more as compared to a jaw crusher.
14. Installation, repair and maintenance cost is less as compared to a gyratory crusher.	Installation, repair and maintenance cost is more as compared to a jaw crusher.
15. It has less flexibility in expansion.	It has more flexibility in expansion.
16. Consumption of power when remain idle require more as compared to a gyratory crusher.	Consumption of power when remain idle require less as compared to a jaw crusher.
17. For crushing rock or ore from particular sized lumps to a particular sized crushed pieces, the receiving opening, capacity, weight, installed power, etc. are less as compared to a gyratory crusher.	For crushing rock or ore from particular sized lumps to a particular sized crushed pieces, the receiving opening, capacity, weight, installed power, etc. are more as compared to a jaw crusher.
18. If the same given quantities can be handled by a single jaw crusher it is more economical.	If the same given quantities is handled by a gyratory crusher, it will be a little bit costlier as compared to jaw crusher.
19. It is more complicated in construction.	It is very simple in construction.

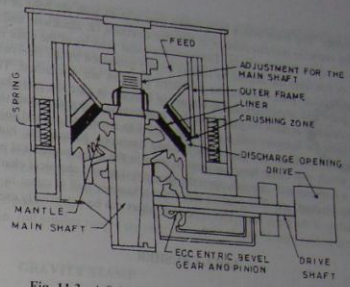


Fig. 11.3 : A Schematic Line Diagram of a Cone Crusher.

- It requires less power per unit of crushed ore or mineral,
- It produces uniform size of the output product,
- Rapid adjustment of the discharge opening is possible.
- Easy to supervise, maintain and repair,
- To avoid clogging ,the feed material should be dry and as far as possible it should be free from fines.
- Due to high speed, the crushed ore or rock particles flow freely.

The reduction ratio varies from 3 to more than 15 although average ratio is 5 to 8. Angle of nip in the short-head machine is less than 10° where as in a standard machine the apparent angle varies from 25° to 30° by corrugating the annular portion of the bowl. The size of the feed is generally less than 85% of the size of the crusher gape. The higher speed as compared to a gyratory crusher, and larger opening of the discharge eliminate the clogging problem. The crushers are rated as per the diameter of the bottom portion of the crushing head.

A 210 cm standard crusher has a weight around 65 t and operated at power varying from 187.5kW to 225 kW. It enable to crush ore or rock ranging from 0.625 cm to 6.25 cm with a discharge capacity ranging from 150t to 900 t. A 90 cm short head cone crusher has weight around 9.5t to 11.25 t and it operates at a power ranging varying from 37 kW to 56 kW. It enables to crush ore or rock ranging from 3 mm to 50 mm with a discharge capacity ranging from 15t to 95t.

HAMMER MILL

It has two units viz (i) a box shape housing (ii) a rotor mounted with hammers. The box shape housing have a circumferentially arranged grates at it's lower part. The grates consists of series of tapered wear resistant steel bars, spaced widely (generally it is above 3mm) depending upon the characteristics of ore. Braker plates are fitted inside the upper part of the housing to assist secondary breaking. These type of plates have replaceable wear resistant lining. The grate opening may be in one set or in several sets to collect different sizes of crushed ore in single operation (Fig. 11.4).

The rotor unit inside the housing contains rows of a hammer (around 8 to 9 hammers per row) which have variety of design (integral head and arm or 2 piece design are available for easy replacement of the hammer head which wear much faster as compared to it's arm), shape and weight depending

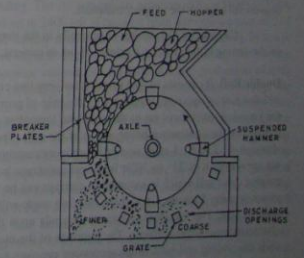


Fig. 11.4 : A Schematic Line Diagram of a Hammer Mill.

upon the capacity of the crusher. The rotor which is mounted over a shaft with the help of two bearings placed at the either end of the housing may rotate in one direction or in both the direction i.e. clockwise and anticlockwise direction to increase the hammer life by using it's both the faces. The circumferential velocity of the rotor is very high which enables to crush the sliding mineral or ore from the hopper by impact energy generated by the free swinging hammer and throw them repeatedly against the breaker plates where the oversized particles are re-crushed. The crushed particles pass through the opening of the grate and accumulated into hopper or hoppers in case the grate have opening in many sets. Some feed materials are also crushed against the grate bars. For maintaining high momentum the hammer mills also have flywheels. The reduction ratio of the hammer mill is very high 20 : 1 to around 50 : 1 and suitable for both the primary and secondary crushing of soft to medium hard mineral or ore like limestone, bauxite, etc. but it should not be used for hard, abrasive ores or rocks which will cause heavy amount of wear in the main mechanical parts and will result severe breakdown. The capacity of a hammer mill depends upon the size, physical and chemical characteristics of the hammer, requirement of the size of the product, etc. The product size from a hammer mill is cubic in nature and the size of the product depends upon the setting of the grate bars. This type of crushers can be extensively used to crush soft to medium hard run of mine ore or mineral to less than 25 mm in size.

ROLL CRUSHER

It is of two types (a) Single roll (b) Double roll.

Single Roll :The machine consists of a moving toothed roll which rotate close to a fixed member of the crushing chamber known as breaker plate. The roll teeth (sluggers)crack down-ward on the feed and compresses the feed against the breaker plate and breaks the feed material (Fig. 11.5).

The pieces which come in between the roll and the breaker plate by the abrasive action are crushed by the dragging pressure and produce heavy amount of dust and fines. The velocity of the moving toothed roll is around 150 m/ sec. This type of crusher are used for yielding coarser production by crushing the laminated, less abrasive , soft to medium hard ore or mineral like coal, gypsum, limestone etc. It does not face any difficulty to crush ore or minerals mixed with clay or sticky materials or the feed in wet condition. Oversized feeds are segregated and recirculated again after sizing them properly. Impact action does better to break friable and laminated ore or mineral. This type of crusher is also useful to crush the slippery materials.

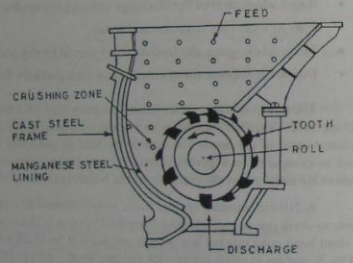


Fig. 11.5 : A Schematic Line Diagram of a Single Roll Crusher.

Capacity of the crusher is generally in the range of 50t to more than 1400 t per hour, which generally depends on the setting of the crusher, hardness of ore or mineral, size of the feed materials, etc. It is a primary crusher.

Double Roll :It consists of two smooth (for very fine crushing), corrugated toothed (for secondary crushing) surface rolls which rotate in opposite direction with the help of prime mover (Fig. 11.6). The feed materials are flown down between the covering shell faces on to the rolls and thereafter are drawn forcefully by the frictional action. Afterward they are crushed by the continuous biting action of the rolls. The ratio of reduction for feed varies from 2 : 1 to more than 10 : 1, although 4 : 1 to 6 : 1 reduction ratio are more common. The reduction of the feed depends upon the setting of angle of nip (generally it is 25° to 30°), frictional resistance between the roll surfaces and the feed material. The driven roll or another drive roll which is mounted in carriage can be slided for adjusting the position of the same either placing it more closer or away from the fixed drive roll (nip angle will also change accordingly) depending upon the requirement of the product size. The frictional resistance depends upon the size, hardness, wet condition, slipperiness of the feed and the characteristics of roll surfaces. The bearings of the movable roll are of heavily spring loaded which are compressed and shift the sliding roll back when excessive pressure caused by the tramp iron and save the crusher from damage. The peripheral speed of the rolls are around 200m/min to more than 350 m/min. The shells are made of either high carbon

steel or of manganese-steel whereas the material of the roll is wear resistant steel casting to withstand the severe shock load caused by the impacts of heavy lumps. High carbon steel can withstand a high value of shock load, cracks slowly, worn-out to very thin section before replacing. Now-a-days replaceable manganese-steel teeth are available to replace the worn out teeth from the rolls.

The roll crushers are very much suitable to crush medium hard materials. The size of the feed may vary from 4cm to 15 cm whereas the product dimension of around 2.5 mm. The roll crusher are very simple, rugged in construction, having lower power and maintenance cost. From very low (30cm x 30 cm) to high capacity rolled crusher (225cm x 90 cm, 225 cm is the roll dia. and 90 cm is the length of the face) single motor or multi-motor drive, from 5 kW to more than 250 kW motor power, from 1.7 t to more than 140 t weight from 28r.p.m to 300 r.p.m. roll crusher ranges are very common.

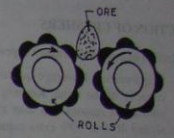


Fig. 11.6 : A Schematic Line Diagram of a Roll Crusher.

GRAVITY STAMP

This type of crusher has high reduction ratio (more than 130) and are used to make finer crushing product of sizes less than 0.5 mm by crushing an average lump size varying from 60mm to 75 mm. The quantity of feed is generally smaller and the gravity stamps are used where no grinding is necessary. It can also be used to crush quartzite ores of gold while amalgamation is necessary. Here crushing is done with the help of 5 heavy stamp (weight around 380kg to 900 kg) togetherly arranged in a single battery (Fig. 11.7). The stamps are raised successively around 12cm to 18 cm with the help of cam and are dropped down (at the rate of around 7 to 8 times/min) to crush the uniformly feed ore laid over the hard metal plate. The ore particles are crushed in aqueous muddy condition and passed through the screen cover the front, rear or both side openings of the mortar box. This type of machine has smaller capacity, occupy huge amount of space and also produces heavy amount of noise and vibration.

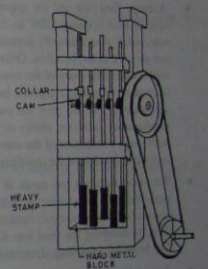


Fig. 11.7 : A Schematic Line Diagram of a Stamp Mill.

AUTOGENOUS GRINDING MILLS

The use of large size grinding mill with hydro cyclone classifier and on line computer for process control have been increasingly in use in large capacity plants. The concept of ore to ore grinding instead of conventional ore to metal grinding has lead to both autogenous and semi-autogenous grinding. The six autogenous grinding mills at Hibbing USA each of 4.6 m x 11.0 m, driven by 2 motors 4500 kW, can grind up to 2400 tph of taconite from nominal top size of 254 mm to 80% -325 mesh, thereby eliminating the secondary and tertiary crushing circuits altogether, resulting in lower capital cost, simpler flow sheet and most importantly the subsequent steps of flotation in controlling the surface properties of mineral. Use of large size hydro cyclone with complete automation will minimize the production of over ground fines and slimes. Since the power required per Tonne of feed in grinding is much greater than crushing, the efficiency of grinding processes is much more significant. Although primary autogenous mills use more energy than conventional grinding system, the saving in total capital costs and significantly lower operating costs compensate for the increased power requirement. Semi autogenous miles and also for lower lining and ball consumption when compared to performance as compared to autogenous miles and also for lower lining and ball consumption when compared to conventional mills. For hard fine grained ore, it is preferred to adopt coarse grinder. This may result in lower grade of concentrate but higher in recovery. The increased cost of metal extraction of this lower grade concentrate may offset the cost of grinding energy reduction but additional recovery would reduce the total energy consumption. Thus an overall cost economics is achieved for concentration at coarser size and hence it is another means of reducing energy requirement. This rejects lot of gangue at coarse size ahead of grinding step. Thus tonnage to be handled in subsequent step is proportionately reduced. Pre-concentration may be done by ore sorter, dense medium gravity separator or magnetic separator. It is claimed that photometric sorting has been successfully applied to wolframite, scheelite ores in

Australia, magnesite ores in Greece and gold ores in South Africa. Other sorters like radiometric, ultraviolet, electrical conductivity and magnetic types are also available.

SELECTION OF CRUSHERS

The selection of a primary crusher depends upon the following factors :

- Requirement of average quantity to be crushed per day or per hour: Indirectly it is related to the production capacity of the mine. Keeping due regards of machine breakdown, stoppage, maintenance, repair and other unforeseen reasons and also availability of sufficient staking place of the crushed output, an allowance of around 40% to 50% extra capacity is added up with the calculated capacity of the crusher. The total amount should be the minimum capacity of the crusher.
- Characteristics of the ore or rock to be handled: Here physical and chemical characteristics of the ore or rock substances like compressive strength, tensile strength, shear strength, plasticity, lithology, abrasivity, moisture content, toughness, stickiness, fracture plane, etc. are very important.
- Average feed size and the requirement of the product size: Depending upon the average larger feed sizes, the size of feed opening (20% to 25% larger than the size of the feed material) and the capacity of a crusher is selected. It also indirectly depends upon the type and size of the loading equipment, size of the fragmentation of ore or rock in the benches. Oversized boulders will cause bridging in the mouth of the crusher and hence they will reduce efficiency of the crusher. Selection of minimum discharge setting value is very important. However instead of selecting a gigantic large high capacity crusher if two nearly half the capacity parallel line crushers are selected , it will give the consistency of production. Multi-crusher system will provide much of machine maintenance facilities, ability to handle from very low production (with the help of only one crusher) to a very high surge production of the mine with minimum operating, maintenance and power cost. It is also to be looked into the reduction of the installation cost, giant building (to suit the crusher dimension) cost, etc. in great depth.
- Transport facilities and mode of feeding in the crusher . It is very important to see that the uniform feeding is provided into the crusher to achieve the best performance and efficiency. Generally the belt conveyor, chain conveyor, plate conveyor, feeder etc. feed the material uniformly into the crusher. Irregular and surge load by directly dumping the feed into the crusher by unloading the rail wagon, trucks etc. will cause a heavy trouble like bridging, chocking, decreasing the capacity and efficiency of crushing, etc.
- Availability of required type and size of indigenous crusher and their proper selection to cope up the spare parts problems.
- During selection of a crusher special attention shall be paid on the following mechanical parts too:
    1. Hydrostat mechanism to raise up and lower down the main shaft assembly to change the crusher setting.
    2. Release of tramp iron
    3. Sophisticated lubrication system
    4. Fixing concaves and mantle
    5. New material for wearing surfaces. Presently alloys with the addition of chromium and nickel are giving better result.
    6. Special crushing chamber
    7. Friction clutches in drive system of jaw crushers
    8. Hydraulic setting arrangement for jaw crushers
    9. Balancing cylinder in the hydraulic system
    10. Prime mover - Selection in between slip ring or squirrel cage induction motor.
- Availability of power supply.
- Availability of capital.
- Environmental condition.
- Cost of the crusher.
- Overall production cost, etc.

Selection of Secondary Crushers

The main factors for selection of secondary crushers are:

- Capacity (may be less than the primary crusher),
- Feed size and product size,
- Material characteristics,
- Operation and maintenance cost,
- Capital expenditure,
- Dependability,
- Maximum economic reduction ratio for each stage by reducing head, number of machines, floor space, etc.
- Open or closed unit design.

Essentials for Selection of a Crusher

- Maximum gap of receiving opening to accommodate the feed easily with no chocking and requires less inspection.
- Maximum discharge opening to permit free passage of final product.
- Feed rate should be uniform. Hence a feeder is to be installed.
- Convenient operation and maintenance,
- Overall robust construction,
- Adequate crushing capability to reduce the hardest and largest lump.
- Non chocking of crushing zone. Finer materials are to be screened out to increase overall efficiency and to prevent chocking.
- Incorporation of safety and protection devices,
- Minimum vibration and noise,
- In place of one big single primary crusher, 2 parallel line crushers are the better proposition.

IN-PIT CRUSHERS

There are various types of in-pit crushers which are as follows :

(a) Near pit crusher-Crusher is located outside the pit but adjacent to the perimeter of the opencast mine.

(b) In-pit crusher-Crusher located within the past or future influence of the ore-body.

(c) Mobile crusher system-Crusher and feeder with integrals propelling mechanism such as walking pads or permanent crawler track units. (Fig. 3.5 and 3.12)

(d) Portable crusher system-Crusher and feeder with independent propelling mechanism such as crawler transporter.

(e) Conventional crusher system-Crusher and feeder housed in reinforced concrete structure which is typically located in large flat area of pit.

A crushing system includes a feed arrangement, a control unit, a crusher and a discharge system. The in-pit crusher should be located as close to the working face as possible. Required safe distance from blasting for a mobile, portable or movable unit is approximately the same as for a shovel. Ore crusher should be located near to the bottom of the pit and waste crushers should be located near to the centre of major overburden removal. The best location for crushers should be identified by quantifying the amounts of ore and waste to be removed from each of the bench levels. In general a greater number of smaller mobile or movable in-pit crushing units are advisable when compared to near pit or in pit conventional crusher system. The reasons for this statement are as follows :

- First the greater number of units allows the units to be kept closer to more working faces.
- The larger crusher impact disproportionately on mine planning when installed in mobile or movable systems because of weight and height parameters.

- In pit crushers suitable for large scale open pit mines, face a couple of problem e.g.
  1. Unit size and unit weight and thus time demand for crusher movement which in case of large units might easily add up to several weeks or months.
  2. Crusher have to be dismantled into a few transportable units.
  3. A tremendous amount of civil works involved for stability or truck access ramps.
  4. Considerable earth movements are necessary.
  5. Belt conveyor shifting or extensions are expensive and time consuming.
  6. In case of hauling of ore, large scale stock piling has to be done.
  7. Maximum truck fleet capacity is in operation just before moving, Minimum truck fleet capacity is required right after the crusher is in its new position.
  8. In case of waste haulage, energy is consumed by dump material which could better be used elsewhere.
  9. The number of dumps then is a matter of purchasable stackers and it is considerably reduced.
  10. The volume of the dump is increased at the same time and thus the haul distance for waste increases.

While the type of machinery used (dipper shovel, wheel loader or trucks) decides the layout of the feed assembly of the mobile crushers, the choice of crusher is governed by the feed boulder size, feed characteristics, required output and ground pressure.

Mobile crushers are in operation with an output of up to 3000tph and a service weight 1000 t.

The choice of transporting mechanism which is given below depends upon soil characteristics and the proposed mining system:

- Hydraulic walking mechanism,
- Crawler track chassis,
- Wheeled (rubber tires) chassis,
- Roll mounted chassis.

Technical specifications of mobile crusher unit

(A) The mobile crusher is a PHB-Weserhutte designed with gyratory crusher type 54/74 and hydraulic walking mechanism fitted with 3 x 200 kW hydraulic pump drives.

Working weight - 1100 t

Installed power - 1300 kW (approx).

Output : 3000 t/h, 0-200/250 mm size. Feeder hopper : 60 m^3 capacity.

Apron feeder :2400 mm wide, centre distance 24 m, incline 25°, 0.08m/s to 0.24 m/s variable speed drive, required power 2 x 200 kW

Spillage chain conveyor:2800 mm wide, centre distance 17.8 m, motor 2.2 kW

Crusher-Allis-Chalmers gyratory crusher type 54/74 ,motor 400 kW, air cooling, 75 kW

Crusher outlet - Apron conveyor - 2600 mm wide centre distance 12.1 m, motor 75 kW.

Spillage conveyor-2800 mm wide, centre distance 9.1 m, motor 2.2 kW.

Mounting Raft :Type coordinated walking mechanism speed up to 0.9m/min, ground pressure when walking 1.5 kg/cm^2, when working 1 kg/cm^2, motors 3 x 200 kW, 1 x 45 kW.

Other features-Maintenance crane, dust suppression unit, etc. are incorporated

Run of mine is loaded into a 60 m^3 feed hopper directly on the apron feeder by a shovel (17 m^3 struck capacity) 19.8 m. boom length with a loading capacity of 2400 t/h of ore/rock.

(B) Mobile crushing and screening unit (manufactured by Voest Alpine, Austria.)

Type STE 10070

The crusher unit has a primary bunker of capacity 15 m^3 where stone block up to one meter long can be loaded. For primary screening it has a vibratory grid-feeder. The unit has STE 10070 Jaw Crusher with a width of 1000mm x 700 mm and has a throughput capacity of 100 t/h. The material is crushed here to a grain size of between 0 and 80 mm. This product then passes to a mobile screening machine where it is separated into grain fractions of 0 to 5 mm, 5mm to 60 mm and 60mm to 80 mm and is thereafter passes to a stockpile. The grain fraction of 5mm to 60 mm is used as the aggregate for steel mills.The other grain sizes are used in the construction industry.

SCREENING

Screening is the process of separation of various uniform sizes (varying from several centimeter to 20 mesh) from a mixture. The screen are also used to feed a particular crusher or grinding mills with particular sizes of particles passed through the screen after eliminating the oversized particles from the mixture. However if the feed material is dry, screening can also be done even below 300 mesh. Screen is made of either parallel welded rods, punched steel plates (punching is done either in round, square or slotted shape) or woven wire cloths. The material of the parallel rod screening surfaces is generally steel rails or steel bars. The punch steel plates are made by punching with dies in circular, slotted or square shape whereas the woven wire cloths are made by gauged wire made from steel, copper or other alloy metals. The gauge wires make square (for coarser particles) or rectangular (for medium size to finer particles) openings. There are many verities of screens are available nowadays . They may be clubbed into two types which are as followings:

(a) **Stationary Screen-Grizzlies** : They are made by parallel welded steel rails or steel bars, placed at an angle (less than the angle of repose of ore or mineral particles) comes under this category (Fig 11.8).The low angle is kept to short out the presence of any metal piece or big boulders before the rejects of the feed enter into the crusher. Moving grizzlies are also available in the market. The grizzlies separate the coarser particles (larger than 25 mm). It has also high capacity, high load bearing capacity, lower price, lower maintenance and operating cost, etc.

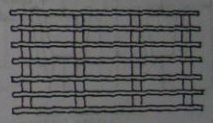


Fig. 11.8 : Grizzly

(b) **Moving Screen** :Under this category the following screens are worth to mention:

(i) **Trommels**-It is a revolving cylindrical or pyramidal shell screen made from the punch steel plate or heavy wire mesh to size the coarser particles which need to wash and disintegrate (e.g. coarser feed particles of gold dredges).

(ii) **Vibrating screen** (Fig. 11.9)-Screening surface of this type of screens are made from woven wire cloth, punched plate or welded bars which is placed at a particular angle and vibration is made either by the rotation of eccentric motors or cams, electromagnetic mechanism or by the hammering action. This type of screens are profoundly used throughout the mining industries. This type of screens has higher screening capacity, lower operating and maintenance cost. Installation is also very easy and cheap. Last but not least is that this type of screens capable of sizing particles of more than 300 mesh.

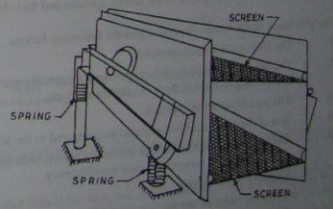


Fig. 11.9 : A Schematic Line Diagram of a Vibratory Screen

(iii) **Shaking screen**-This type of screens shake longitudinally instead of vibrating and feed particles are subjected to shock loads at higher amplitude. The screen is placed at flexible attachments which is shaken by the rods or other eccentric mechanism.

The screen is either placed horizontally or at an angle around 17°. This type of screen is subjected to heavy vibrations which adversely affect the supporting structure. Maintenance cost is heavy in this type of screen.

Screening Efficiency-It is generally a measure of effectiveness of a screening operation relative to the perfect screening operation. In broad terms it may be defined as the ratio of the fine materials or grains those are passed through the screen from the feed to the quantity of the fines

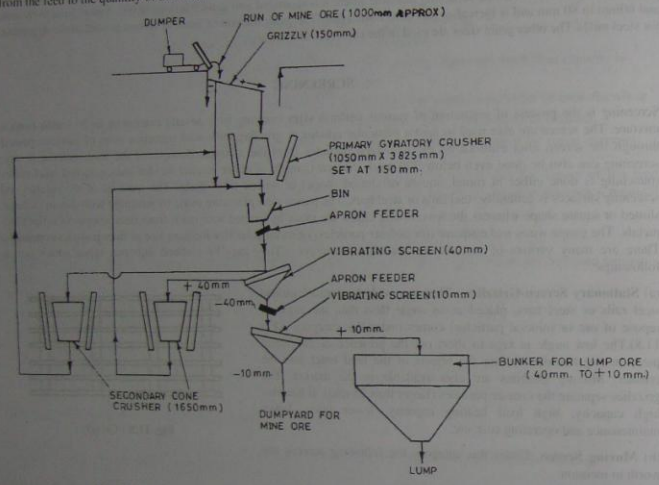


Fig. 11.10 : A Generalised Crushing and Screening Flow Sheet of Run of Mine Ore.

present in the feed material. This definition is taken due consideration to separate coarser materials from the mixture of both fines and coarser material which are feed into the screen but it does not take due accounts of the grains which are just smaller in size as compared to the screen aperture and they do not passed through the screen.

The efficiency of the screen depends upon the following factors:

1. The size of the screen aperture.
2. The physical characteristics of ore like size, specific gravity, stickiness, moisture content and watery condition, amount of presence of fines or clay material, etc.
3. The area of screening surface.
4. Size of the feed particles or grains as compared to the screen aperture.
5. The screening mechanism like stationary screen, shaking screen, vibrating screen etc. Amount of more movement of the screens increase the efficiency.
6. Capacity of the screen.
7. The percentage of opening (or availability of opening) to the total surface in the screen.
8. The angular orientation of the screen (angle from the vertical plane must be more than the angle of repose of the material).

9. The velocity of the flow of feed in the screen.
10. Efficiency of operation and maintenance.

The capacity of the screen also depends upon the above mentioned factors (except the point no-6).The cost of the screening plant depends upon the cost of the screen and the bulky building to accommodate the total screening circuit .It has also to take due consideration of free falling of ore over the screen. The cost includes installation cost, operating and maintenance cost, cost of supervision, power cost (for moving screen), etc.

Decrease of Screening Efficiency and Improvements

The following are the main reasons of decreasing the efficiency of screening and the proposed measures for improvement :

1. During the selection of the screen it would be noted that the aperture of the screen should be optimum in size depending upon the size of the feed and the requirement of the size of the screened particles. The number of apertures per unit surface area is very important. The efficiency and the capacity is directly proportional to the size of the aperture. Because of prolonged operation, feed materials may clog the screen apertures which will decrease the screening area and hence the screening efficiency. Therefore regular supervision, cleaning of the screen surface and also it's maintenance is very important.

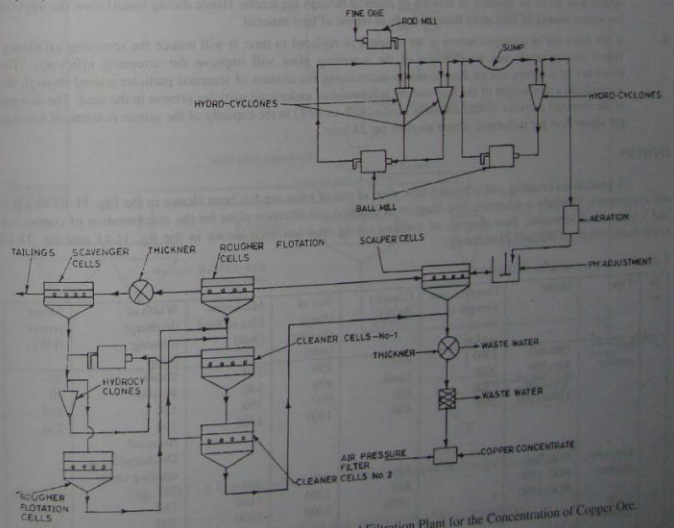


Fig. 11.11 : A Schematic Line Diagram of Flotation and Filtration Plant for the Concentration of Copper Ore.

2. Size of the feed material (particles of ore or rock) should be such that only sufficiently smaller sized particles should pass through the screen whereas huge amount of particles (just a little bit smaller size relative to the aperture of the screen) generally do not pass through it. Hence efficiency decreases. This problem can be solved during the extraction of rocks or mineral by improved processes.

3. Stickiness, presence of moisture and watery condition in the feed particles, presence of clay and other fine materials along with water hinder the efficiency of the screen; since under these circumstances materials try to stick with each other and clog the opening of the aperture. Drying of ore before screening and use of moving screen, avoiding of loading of fines or clay materials in the transport system which feeds the crusher, proper maintenance, etc. will solve these problems.
4. It is already mentioned that the moving screen (vibrating screen, shaking screen, etc.) has higher efficiency as compared to the stationary screen. More the amount of movement more will be the screening efficiency. Hence if the efficiency of the moving mechanism goes down, the efficiency of the screening mechanism also decreases. Hence selection of proper movement of the moving mechanism and proper maintenance system will improve the screening efficiency.
5. Capacity of the screen is also related to the efficiency of the system. Screening efficiency can generally be improved if the screen is utilized below its maximum capacity.
6. The screening efficiency will be decreased if the velocity of the feed material into the screen is very low. Hence a particular height of falling of the feed material is to be maintained to improve the particle velocity into the screen which will improve the screening efficiency.
7. Screening efficiency will decrease if the angle of inclination of the screen from the vertical plane is less as compared to the angle of repose of the feed material. The accumulated rejects (over sized particle) over the screen will act as an obstacle of flowing of material through the screen. Hence during installation the angle of the screen should be kept more than the angle of repose of feed material.
8. If the worn out or damaged screen is not repaired or replaced in time, it will reduce the screening efficiency. Hence proper repair and maintenance of the screening plant will improve the screening efficiency. The efficiency of a screen can be determined by ascertaining the amount of screened particles passed through the screen from a unit weight of the feed and the percentage of undersized particles present in the feed. The amount of the screened particles should resemble nearly (80% to 90%) to the capacity of the screen in terms of tonnage per square foot per millimeter screen aperture per 24 hour.

OTHERS

A generalized crushing and screening flow sheet of run of mine ore has been shown in the Fig. 11.10 which is self explanatory. Similarly a schematic line diagram of flotation and filtration plant for the concentration of copper ore and a generalized schematic flow diagram of washing coking coal has been shown in the fig. 11.11 and fig. 11.12 respectively which are also self explanatory.

Table 11.1 A few examples of crushers

Sl. no	Type	Model	Compres-sive strength of material (kg/cm^2)	Capacity (M^3/h)	Size of charge opening (mm)	Maximum Size of feed (mm)	Width of discharge opening (mm)	Motor power (kW)
1	Jaw crusher	250x400 400x900 900x1200 1500x2100	2500 3000 2500 2500	3.5-12 20-48 130 450	250 400 900 1500	210 340 750 1300	20-80 40-100 130 180	20 40 100 250
2	Gyratory crusher	PGC-500 PGC-1200 PGC-1500	- - -	150 800 1450	500 1200 1500	400 1100 1300	Nominal Discharge set at opening (mm) 75 180 200	125 2x200 2x320
3	Cone crusher							
	Standard	SCC1200A SCC2200B	- -	30-85 340-580	115 350	100 300	Width of discharge opening (mm) 8-25 30-60	75 250
	Short-head	SHC1200 SHC2200	- -	12-55 75-220	45 130	38 100	3-13 5-15	75 250

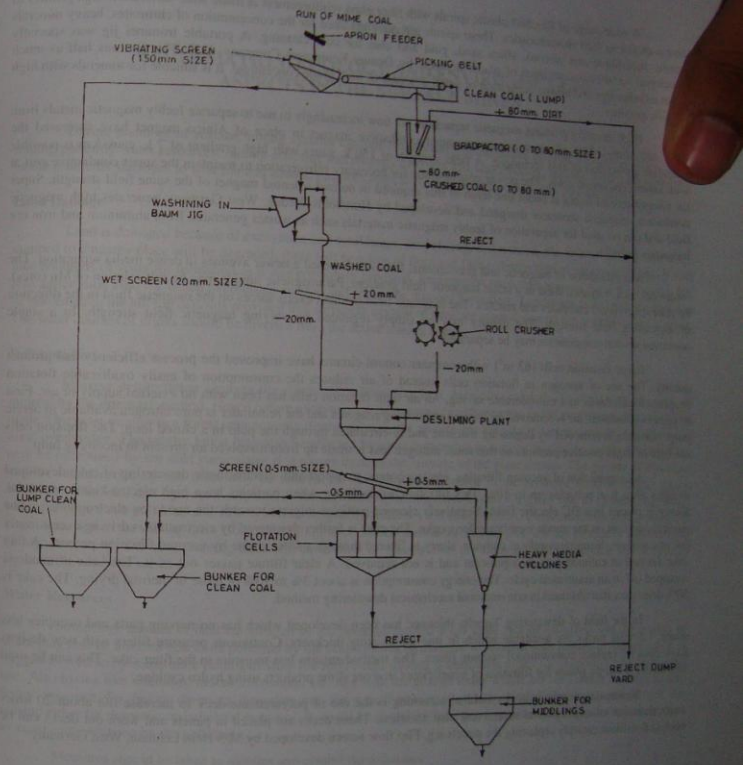
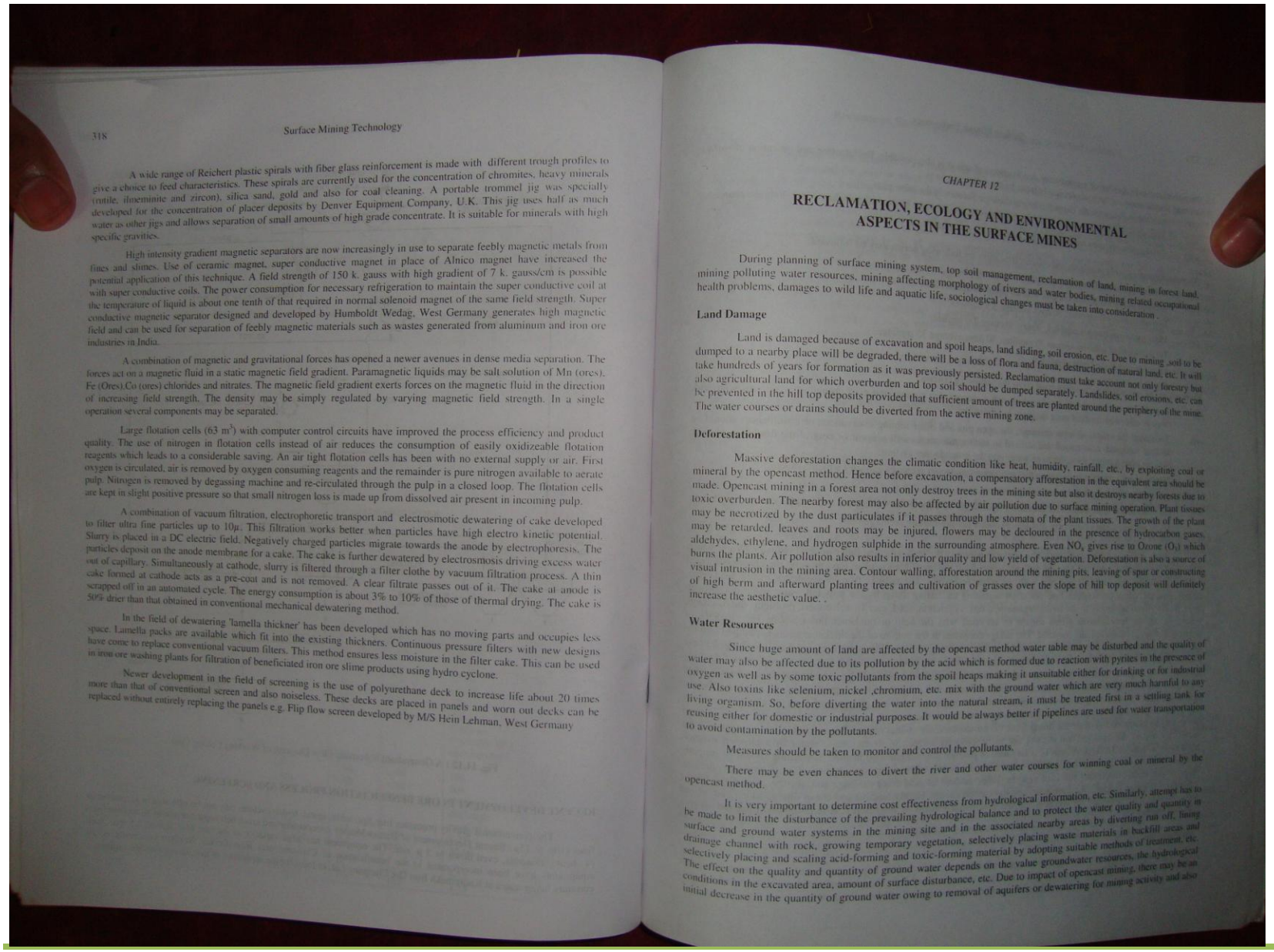


Fig. 11.12 : A Generalised Schematic Flow Diagram of Washing Coking Coal

RECENT DEVELOPMENT IN ORE BENEFICIATION PROCESS AND SCREENING

The conventional gravity processes like jigs, table, spiral, hydro cyclone may not be effective in separation of fines (10μ to 15μ).Recent Development of Barties-Mozley concentrator and cross belt separator allows to recover fines of heavy minerals, even down to 5 μ size. The equipment has large capacity at the rate of 1.5tph to 3.5 tph. Such equipments have been incorporated in the beneficiation of Chromite ores from Orissa. Reichert concentrators are currently being tested at Kudremukh Iron Ore Company Ltd. in India for the separation of iron values from the tailings.



A wide range of Reichert plastic spirals with fiber glass reinforcement is made with different trough profiles to give a choice to feed characteristics. These spirals are currently used for the concentration of chromites, heavy minerals (rutile, ilmeminite and zircon), silica sand, gold and also for coal cleaning. A portable trommel jig was specially developed for the concentration of placer deposits by Denver Equipment Company, U.K. This jig uses half as much water as other jigs and allows separation of small amounts of high grade concentrate. It is suitable for minerals with high specific gravities.

High intensity gradient magnetic separators are now increasingly in use to separate feebly magnetic metals from fines and slimes. Use of ceramic magnet, super conductive magnet in place of Alnico magnet have increased the potential application of this technique. A field strength of 150 k. gauss with high gradient of 7 k. gauss/cm is possible with super conductive coils. The power consumption for necessary refrigeration to maintain the super conductive coil at the temperature of liquid is about one tenth of that required in normal solenoid magnet of the same field strength. Super conductive magnetic separator designed and developed by Humboldt Wedag, West Germany generates high magnetic field and can be used for separation of feebly magnetic materials such as wastes generated from aluminum and iron ore industries in India.

A combination of magnetic and gravitational forces has opened a newer avenues in dense media separation. The forces act on a magnetic fluid in a static magnetic field gradient. Paramagnetic liquids may be salt solution of Mn (ores), Fe (Ores),Co (ores) chlorides and nitrates. The magnetic field gradient exerts forces on the magnetic fluid in the direction of increasing field strength. The density may be simply regulated by varying magnetic field strength. In a single operation several components may be separated.

Large flotation cells (63 m^3) with computer control circuits have improved the process efficiency and product quality. The use of nitrogen in flotation cells instead of air reduces the consumption of easily oxidizeable flotation reagents which leads to a considerable saving. An air tight flotation cells has been with no external supply or air. First oxygen is circulated, air is removed by oxygen consuming reagents and the remainder is pure nitrogen available to aerate pulp. Nitrogen is removed by degassing machine and re-circulated through the pulp in a closed loop. The flotation cells are kept in slight positive pressure so that small nitrogen loss is made up from dissolved air present in incoming pulp.

A combination of vacuum filtration, electrophoretic transport and electrosmotic dewatering of cake developed to filter ultra fine particles up to 10μ. This filtration works better when particles have high electro kinetic potential. Slurry is placed in a DC electric field. Negatively charged particles migrate towards the anode by electrophoresis. The particles deposit on the anode membrane for a cake. The cake is further dewatered by electrosmosis driving excess water out of capillary. Simultaneously at cathode, slurry is filtered through a filter clothe by vacuum filtration process. A thin cake formed at cathode acts as a pre-coat and is not removed. A clear filtrate passes out of it. The cake at anode is scrapped off in an automated cycle. The energy consumption is about 3% to 10% of those of thermal drying. The cake is 50% drier than that obtained in conventional mechanical dewatering method.

In the field of dewatering 'lamella thickner' has been developed which has no moving parts and occupies less space. Lamella packs are available which fit into the existing thickners. Continuous pressure filters with new designs have come to replace conventional vacuum filters. This method ensures less moisture in the filter cake. This can be used in iron ore washing plants for filtration of beneficiated iron ore slime products using hydro cyclone.

Newer development in the field of screening is the use of polyurethane deck to increase life about 20 times more than that of conventional screen and also noiseless. These decks are placed in panels and worn out decks can be replaced without entirely replacing the panels e.g. Flip flow screen developed by M/S Hein Lehman, West Germany

CHAPTER 12

RECLAMATION, ECOLOGY AND ENVIRONMENTAL ASPECTS IN THE SURFACE MINES

During planning of surface mining system, top soil management, reclamation of land, mining in forest land, mining polluting water resources, mining affecting morphology of rivers and water bodies, mining related occupational health problems, damages to wild life and aquatic life, sociological changes must be taken into consideration .

Land Damage

Land is damaged because of excavation and spoil heaps, land sliding, soil erosion, etc. Due to mining ,soil to be dumped to a nearby place will be degraded, there will be a loss of flora and fauna, destruction of natural land, etc. It will take hundreds of years for formation as it was previously persisted. Reclamation must take account not only forestry but also agricultural land for which overburden and top soil should be dumped separately. Landslides, soil erosions, etc. can be prevented in the hill top deposits provided that sufficient amount of trees are planted around the periphery of the mine. The water courses or drains should be diverted from the active mining zone.

Deforestation

Massive deforestation changes the climatic condition like heat, humidity, rainfall, etc., by exploiting coal or mineral by the opencast method. Hence before excavation, a compensatory afforestation in the equivalent area should be made. Opencast mining in a forest area not only destroy trees in the mining site but also it destroys nearby forests due to toxic overburden. The nearby forest may also be affected by air pollution due to surface mining operation. Plant tissues may be necrotized by the dust particulates if it passes through the stomata of the plant tissues. The growth of the plant may be retarded, leaves and roots may be injured, flowers may be decloured in the presence of hydrocarbon gases, aldehydes, ethylene, and hydrogen sulphide in the surrounding atmosphere. Even NO_x gives rise to Ozone (O_3) which burns the plants. Air pollution also results in inferior quality and low yield of vegetation. Deforestation is also a source of visual intrusion in the mining area. Contour walling, afforestation around the mining pits, leaving of spur or constructing of high berm and afterward planting trees and cultivation of grasses over the slope of hill top deposit will definitely increase the aesthetic value. .

Water Resources

Since huge amount of land are affected by the opencast method water table may be disturbed and the quality of water may also be affected due to its pollution by the acid which is formed due to reaction with pyrites in the presence of oxygen as well as by some toxic pollutants from the spoil heaps making it unsuitable either for drinking or for industrial use. Also toxins like selenium, nickel ,chromium, etc. mix with the ground water which are very much harmful to any living organism. So, before diverting the water into the natural stream, it must be treated first in a settling tank for reusing either for domestic or industrial purposes. It would be always better if pipelines are used for water transportation to avoid contamination by the pollutants.

Measures should be taken to monitor and control the pollutants.

There may be even chances to divert the river and other water courses for winning coal or mineral by the opencast method.

It is very important to determine cost effectiveness from hydrological information, etc. Similarly, attempt has to be made to limit the disturbance of the prevailing hydrological balance and to protect the water quality and quantity in surface and ground water systems in the mining site and in the associated nearby areas by diverting run off, lining drainage channel with rock, growing temporary vegetation, selectively placing waste materials in backfill areas and selectively placing and sealing acid-forming and toxic-forming material by adopting suitable methods of treatment, etc. The effect on the quality and quantity of ground water depends on the value groundwater resources, the hydrological conditions in the excavated area, amount of surface disturbance, etc. Due to impact of opencast mining, there may be an initial decrease in the quantity of ground water owing to removal of aquifers or dewatering for mining activity and also

an alteration in the quality of groundwater due to chemical action is also possible. Reclamation and operation of surface mining should maintain the hydrological balance by the following ways:

(i) Minimizing changes for the prevailing hydrologic balance in the mine plan.

(ii) Minimizing changes in water quality and quantity of groundwater by diverting surface water drainage channels.

(iii) Taking precautionary measures during operation to minimize water pollution.

To maintain water quality standards and effluent limitation the following norms may be followed:

(i) All drainage should pass through sedimentation ponds.

(ii) Effluent limitation should comply with all state and federal laws

(a) Iron-maximum 7.0 mg/liter, 30 days average 3.5 mg/liter.

(b) Manganese-maximum 4 mg/liter, 30days average 2.0 mg/liter.

(c) Total suspended solid-maximum 70 mg/liter, 30 days average 35 mg/liter.

(d) pH value-within the range of 6.0-9.0.

Plans related to groundwater protection must show the following criteria:

(i) Back filling material should be placed in such a manner that contamination of groundwater systems with acid, toxic or otherwise harmful mine drainage is minimized.

(ii) To control the effects of mine drainage, the open pits and other mining excavations or disturbances should be located, designed, constructed and utilized in such a manner as will prevent or control the discharge of acid, toxic or other harmful mine drainage water into the groundwater system.

(iii) Mining should result in reclamation that restores groundwater recharge capacity which would

(a) support the approved post mining land use,

(b) minimize disturbances for prevailing hydrologic balance in the mining site and in adjacent areas and

(c) provide a rate of recharge that approximates the pre-mining recharge rate.

Water pollution problem is more destructive in nature in the lead, zinc and other mineral industry. Water which is pumped out from the mine and also the polluted water from the beneficiation plant pass through the channels formed by the soil erosion and wash off and mix with the water reservoirs. The tailings from concentrator plants, lead, zinc, copper, gold etc. cause tremendous land pollution adjoining the tailing dump by the polluted water come out through them. It also affects adversely to the living organism, crops, vegetation, food, etc. To solve these problems the discharge water from all the beneficiation plants should be recycled with the help of thickener tanks where 75% to 85% of discharged water is recovered and recycled. The sludge deposited at the bottom of the thickener tanks is pumped into the tailing pond where the solid materials get settled and the remaining 15% to 25% water is cleaned and discharged to the natural stream of water. Quality of the discharged water should be constantly monitored. Construction of earth fill tailing dam for tailing disposal from the concentrator plants and coursing of water through the cemented drainage channel fitted with check dams at regular intervals to arrest the solid sediments will solve the siltation problem of river bed and also the agriculture land . Compacting of spoil dumps can be done by covering it with the help of layer of clay extracted out during development of the mine. Sometime even layer of clay in the form of sandwich in between the layers of spoil dump may be practiced to isolate the dump from the atmospheric air so that oxygen can not come into contact with the spoils. To protect the land erosion ditches are to be properly located to divert mine water in a planned manner, advantages of natural drainage system is to be taken in planned manner and also method of working should be properly selected. Replenishment of top soil, grading towards the slope at a mild inclination, planting of grass, trees, etc. are also some of the means to control the land erosion. Huge amount of acid mine drainage can also be neutralized by the addition of alkaline reagent like limestone, sodium carbonate, etc. Highly reactive and cheap lime is added and aerated with the acid mine drainage for reaction and the precipitant yielded by the reaction is removed and disposed off in a planned manner.

Air Pollution

Air pollution is due to presence of contaminants in air such as dust, fumes, gases, mist, odor, smoke or vapor in such a quantities and of characteristics for a particular duration which may injurious to human, plant or animal life or to property or which unreasonably interfere with comfortable environment of life and property.

Air pollution is not necessarily be judged by the weight proportion alone of the pollutant but also with its nature, concentration and the point at which it is emitted. The sources of air pollution (by dust particulates) are mainly due to drilling, blasting, loading and unloading of coal, mineral and overburden, transport of material, movement of heavy vehicle, coal handling plant, large open surface and large overburden surface, water disposal, land reclamation, etc. Smoke and harmful gases pollution are due to fire in coal stacks, domestic use of coal, open bhattas, underground fire of coal, auto-oxidation and release of gases from coal and overburden strata, etc. Oxides of sulfur, carbon, nitrogen and hydrocarbon are considered to be of serious atmospheric pollutants. Mining equipment like shovels, trucks, compressors, dozers and dumpers run by diesel oil also produce this type of pollutant. Five liters of gasoline burnt by vehicles and other machines produce nearly 1.5 kg. of CO sufficient to pollute nearly 2265 m^3 air to the threshold limit and 0.02-0.07 kg of nitrogen oxides are sufficient to pollute 5664 m^3 of air to the threshold limit. Switching over from diesel driven engines to the electric driven engines, control of fire in the underground coal seams, reduction in the consumption rate of explosives, controlling of surface fire in the coal dumps, reducing the domestic use of coal, switching over from open bhattas to the special smokeless fuel technology for producing soft coke are some of the controlling measures for gaseous pollution.

The reduction of SO_2 in the ambient air is possible by using low sulfur fuels and by fuel gas desulphurization whereas the formation of nitrogen gases during combustion processes can be reduced by fuel modification or changing the combustion condition.

Dust particles are generally categorized into two types:

(i) settleable type (sizes more than 100 μm)

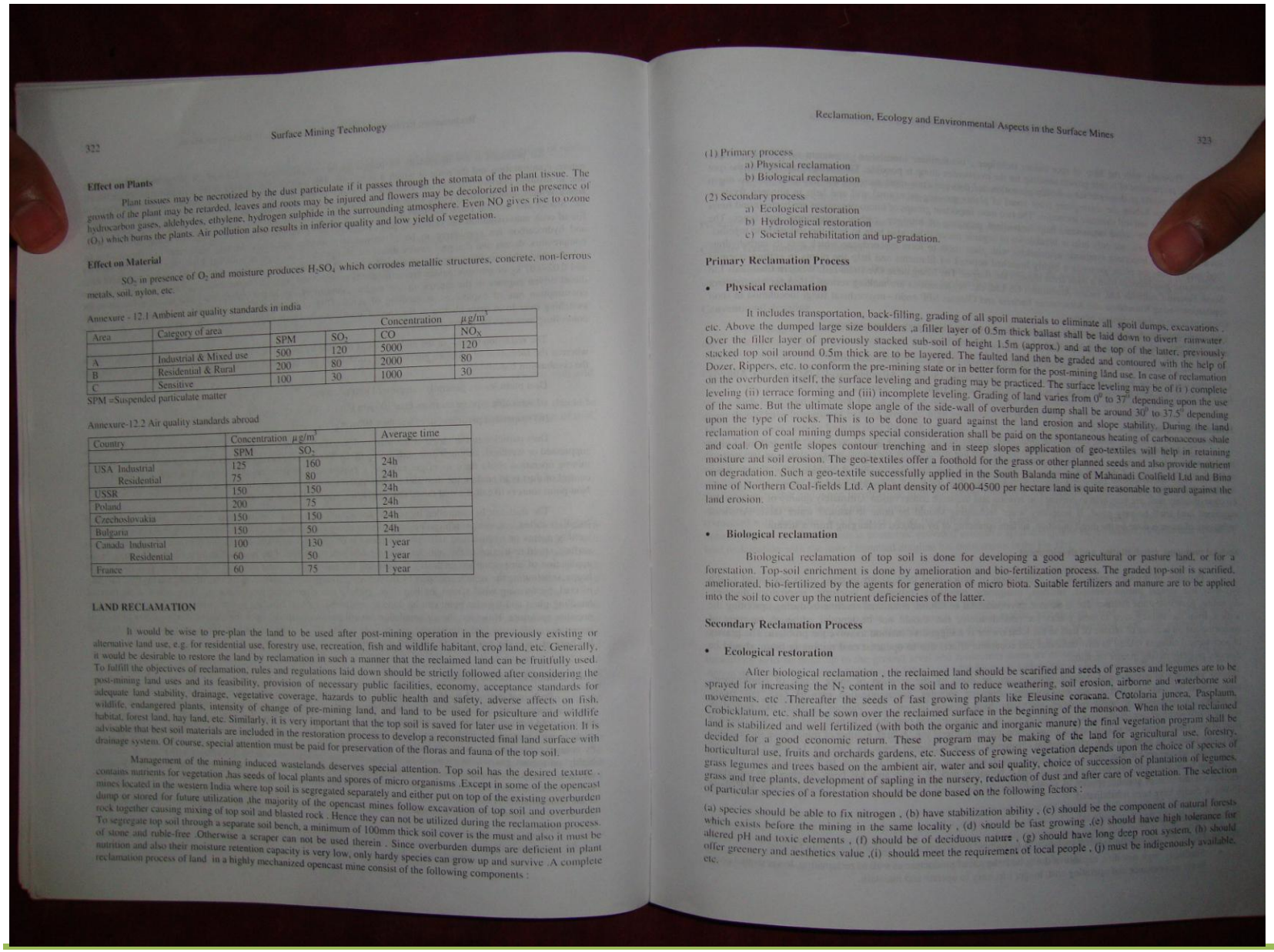
(ii) suspended particles (sizes less than 100 μm)

Dust particles in the range of 0.5 μm to 5 μm sizes are very dangerous to the human system and have to be suppressed or screened, although dust particles up to the 300μm range is very important. The dust that pollutes air due to mining operation could have its origin either in a point source or in a non-point source. The basic philosophy in the control of dust is to contain the dust and to convert the non-point emissions to point source (e.g, chimney) for disposal. Non-point sources like dust cloud raised during blasting is quite difficult to control.

The dust can be controlled by regular application of water in the haul road, stabilizing chemically (chemical stabilizer called 'coherex' which can be applied after every six weeks in the ratio of coherex : water = 1:10) the road by wetting agents or hygroscopic salts, prompt removal of coal and rock and compacting the road to stabilize the road surface, speed restriction of the vehicles, re-vegetation near to the site of the road, wet drilling, restricted blasting, by the application of dust scrubber, enclosing or covering the rail wagon, water spraying on the dumper, coal stacks, spoil heaps, substituting the vehicle transport by the conveyor transport, by minimizing the area of disturbed land and handling of coal, by forming wind screen, by bag filter, cyclone and electrostatic precipitator etc. Dust control at the material handling plant and transfer point can be done by improved material handling system, water spraying and by negative pressure enclosure. However, the air population control measures depends upon the climatic condition, size and type of operation, quality standard requirement and existing quality. The standards of air quality in India and stipulated abroad has been given in the Annexure 12.1 to 12.2 respectively.

Air Pollution Effects on Human

Both coal dust and stone dust in the range of 0.5 μm to 5 μm sizes are responsible for pneumoconiosis and silicosis, respectively. The threshold limit of airborne respirable dust has been fixed at 3 mg/m^3 by the DGMS, whereas the threshold limit of airborne respirable dust in the USA is 2 mg/m^3. Carbon monoxide is a deadly poisonous gas which reacting with hemoglobin in the human blood converts it to carboxyl hemoglobin and even at the concentration of 0.1% by volume causes death to a person within 4h. Threshold limit of SO_2 concentration is 5 ppm. Hydrogen sulphide is a highly poisonous gas whose presence at the concentration of 0.07 % to 0.2% by volume will result in the rapid unconsciousness of a human and death in a few minutes at higher concentrations. It's threshold limit is 10 ppm. Similarly sulfur dioxide is also a very poisonous gas which is extremely irritating to the eyes and attacks respiratory passages even at the concentration of 0.002% by volume. Similarly nitrous fumes are also highly poisonous and at the concentration rate of 700 ppm causes death to a person after short exposure. The threshold limit of concentration of NO =25 ppm and NO_2 =5 ppm.



Effect on Plants

Plant tissues may be necrotized by the dust particulate if it passes through the stomata of the plant tissue. The growth of the plant may be retarded, leaves and roots may be injured and flowers may be decolorized in the presence of hydrocarbon gases, aldehydes, ethylene, hydrogen sulphide in the surrounding atmosphere. Even NO gives rise to ozone (O_3) which burns the plants. Air pollution also results in inferior quality and low yield of vegetation.

Effect on Material

SO_2 in presence of O_2 and moisture produces H_2SO_4 which corrodes metallic structures, concrete, non-ferrous metals, soil, nylon, etc.

Annexure - 12.1 Ambient air quality standards in india

Area	Category of area	Concentration $\mu g/m^3$			
		SPM	SO_2	CO	NO_X
A	Industrial & Mixed use	500	120	5000	120
B	Residential & Rural	200	80	2000	80
C	Sensitive	100	30	1000	30

SPM =Suspended particulate matter

Annexure-12.2 Air quality standards abroad

Country	Concentration $\mu g/m^3$		Average time
	SPM	SO_2	
USA Industrial	125	160	24h
Residential	75	80	24h
USSR	150	150	24h
Poland	200	75	24h
Czechoslovakia	150	150	24h
Bulgaria	150	50	24h
Canada Industrial	100	130	1 year
Residential	60	50	1 year
France	60	75	1 year

LAND RECLAMATION

It would be wise to pre-plan the land to be used after post-mining operation in the previously existing or alternative land use, e.g. for residential use, forestry use, recreation, fish and wildlife habitant, crop land, etc. Generally, it would be desirable to restore the land by reclamation in such a manner that the reclaimed land can be fruitfully used. To fulfill the objectives of reclamation, rules and regulations laid down should be strictly followed after considering the post-mining land uses and its feasibility, provision of necessary public facilities, economy, acceptance standards for adequate land stability, drainage, vegetative coverage, hazards to public health and safety, adverse affects on fish, wildlife, endangered plants, intensity of change of pre-mining land, and land to be used for psiculture and wildlife habitat, forest land, hay land, etc. Similarly, it is very important that the top soil is saved for later use in vegetation. It is advisable that best soil materials are included in the restoration process to develop a reconstructed final land surface with drainage system. Of course, special attention must be paid for preservation of the floras and fauna of the top soil.

Management of the mining induced wastelands deserves special attention. Top soil has the desired texture , contains nutrients for vegetation ,has seeds of local plants and spores of micro organisms .Except in some of the opencast mines located in the western India where top soil is segregated separately and either put on top of the existing overburden dump or stored for future utilization ,the majority of the opencast mines follow excavation of top soil and overburden rock together causing mixing of top soil and blasted rock . Hence they can not be utilized during the reclamation process. To segregate top soil through a separate soil bench, a minimum of 100mm thick soil cover is the must and also it must be of stone and ruble-free .Otherwise a scraper can not be used therein . Since overburden dumps are deficient in plant nutrition and also their moisture retention capacity is very low, only hardy species can grow up and survive .A complete reclamation process of land in a highly mechanized opencast mine consist of the following components :

(1) Primary process
- a) Physical reclamation
- b) Biological reclamation

(2) Secondary process
- a) Ecological restoration
- b) Hydrological restoration
- c) Societal rehabilitation and up-gradation.

Primary Reclamation Process

- **Physical reclamation**

It includes transportation, back-filling, grading of all spoil materials to eliminate all spoil dumps, excavations , etc. Above the dumped large size boulders ,a filler layer of 0.5m thick ballast shall be laid down to divert rainwater. Over the filler layer of previously stacked sub-soil of height 1.5m (approx.) and at the top of the latter, previously stacked top soil around 0.5m thick are to be layered. The faulted land then be graded and contoured with the help of Dozer, Rippers, etc. to conform the pre-mining state or in better form for the post-mining land use. In case of reclamation on the overburden itself, the surface leveling and grading may be practiced. The surface leveling may be of (i) complete leveling (ii) terrace forming and (iii) incomplete leveling. Grading of land varies from 0^0 to 37^0 depending upon the use of the same. But the ultimate slope angle of the side-wall of overburden dump shall be around 30^0 to 37.5^0 depending upon the type of rocks. This is to be done to guard against the land erosion and slope stability. During the land reclamation of coal mining dumps special consideration shall be paid on the spontaneous heating of carbonaceous shale and coal. On gentle slopes contour trenching and in steep slopes application of geo-textiles will help in retaining moisture and soil erosion. The geo-textiles offer a foothold for the grass or other planned seeds and also provide nutrient on degradation. Such a geo-textile successfully applied in the South Balanda mine of Mahanadi Coalfield Ltd and Bina mine of Northern Coal-fields Ltd. A plant density of 4000-4500 per hectare land is quite reasonable to guard against the land erosion.

- **Biological reclamation**

Biological reclamation of top soil is done for developing a good agricultural or pasture land, or for a forestation. Top-soil enrichment is done by amelioration and bio-fertilization process. The graded top-soil is scarified, ameliorated, bio-fertilized by the agents for generation of micro biota. Suitable fertilizers and manure are to be applied into the soil to cover up the nutrient deficiencies of the latter.

Secondary Reclamation Process

- **Ecological restoration**

After biological reclamation , the reclaimed land should be scarified and seeds of grasses and legumes are to be sprayed for increasing the N_2 content in the soil and to reduce weathering, soil erosion, airborne and waterborne soil movements, etc .Thereafter the seeds of fast growing plants like Eleusine coracana, Crotolaria juncea, Pasplaum, Crobicklatum, etc. shall be sown over the reclaimed surface in the beginning of the monsoon. When the total reclaimed land is stabilized and well fertilized (with both the organic and inorganic manure) the final vegetation program shall be decided for a good economic return. These program may be making of the land for agricultural use, forestry, horticultural use, fruits and orchards gardens, etc. Success of growing vegetation depends upon the choice of species of grass legumes and trees based on the ambient air, water and soil quality, choice of succession of plantation of legumes, grass and tree plants, development of sapling in the nursery, reduction of dust and after care of vegetation. The selection of particular species of a forestation should be done based on the following factors :

(a) species should be able to fix nitrogen , (b) have stabilization ability , (c) should be the component of natural forests which exists before the mining in the same locality , (d) should be fast growing ,(e) should have high tolerance for altered pH and toxic elements , (f) should be of deciduous nature , (g) should have long deep root system, (h) should offer greenery and aesthetics value ,(i) should meet the requirement of local people , (j) must be indigenously available, etc.

Presently with the help of root trainer technique , bio-fertilizer inoculation treatment or by tissue culture technique, preparation of higher growth seedling for the overburden dump is possible. The multiple tap roots of the root trained plant would go deeper in the soil after plantation and thus can draw more moisture and nutrients. Various micro organism groups habitat the rhizosphere (root zone) of plants growing on normal soil and help in nitrogen fixation, nutrient recycling and soil structure formation. The two most important groups of micro-organisms are bacteria and fungi .Bacteria, the single celled organisms, help in nutrient recycling of nitrogen ,carbon ,phosphorus and fungi. The filamentous organisms, not only help in breakdown of organic material but also help to held the soil particle together. Some fungi make important symbiotic associations with plant roots to form mycorrhizae which gets its carbohydrate food supply from the plant, grows in a highly branched network of filaments and help the plants in getting essential nutrients like phosphorous from a wide areas through the fungus. The Manganese Ore India Ltd, Western Coalfields Ltd ,South Eastern Coalfields Ltd, Indian Aluminium Co Ltd, etc. organizations are making serious effort for bio-fertilizer application –using Rhizobium and Azotobacter bacteria and Glomus SPP endo –mycorrhizal fungi inoculation in spoil dumps of some of their opencast mines.

The tissue culture makes production of exact copies of trees selected for superior characteristics.

A plant density of 2000-2500 per hectare land is quite good.

The following species found to have reasonably higher growth on the leveled overburden dump :

Causarina equisetifolia, Eucalyptus, Cassia sciamea, Delbergia sissoo (Sisham), Ailanthus excelsa (Maheneem), Albizzia lebbeck (Siris), Pongamia pinnata ((Karanj), Acacia nilotica (Babool), etc.

Hydrological restoration

The hydrology of the reclaimed area shall be as close as to the original state as possible. The soil must have enough water holding capacity. With the adequate plantation of trees, grasses and legumes, the water run-off shall be reduced and the quality of the effluents from the rehabilitation area shall be improved considerably. Beside these adequate engineering measures like bench terracing ,construction of check dams ,boulder pitching ,plugging of gullies, erection of safety wall ,etc may help to prevent soil erosion and moisture conservation .Ultimately quality of water in the reclaimed land shall be very good. If necessary water recharging should be done to natural water table. Artificial recharging of water is done by the direct injection, surface spreading or by induced recharging from a stream.

Societal problems, rehabilitation and development

Pollution of drinking water and air, degradation of agricultural land, sharp increase in occupational health problems like miners pneumoconiosis, silicosis, manganese poisoning, asbestosis, siderosis, respiratory diseases, dermatitis, gastrointestinal diseases due to adverse environmental condition, emotional sentiments during uprooting the settlers from the mining area to new places for rehabilitations, etc. should not be ignored, although payment of compensation on account of seizure of land to the local owner is a suggestive method to solve the problem. The gravity of sociological changes related to environmental and economic effects due to opencast coal mining in future should be assessed properly paying greater attention.

Uplifting of society and cultural heritage can be done by the creation of more scope of employment, establishment of more number of schools, colleges, shopping complex, communication network (viz. rail, roads, telephones), up-gradation of health facilities, electricity, high average income, etc. Various facilities like recreation facilities, marketing center, clubs, parks, townships, etc. may be constructed over the reclaimed land. The most common form of post mining land use in India is the afforestation and agricultural use. A large number of mines have successfully reclaimed excavated land through afforastation. However many reclaimed areas in Neyveli Lignite Corporation, Jhingurdah opencast coal mines, etc have been converted to parks and recreational facilities. In Noamundi iron ore mine in addition to reclamation of 105 ha excavated land a botanical park of 18 ha area has been developed to undertake research on selection of suitable species of plants for different types of land. In Bhadanpur limestone mine, M.P about 15.9ha of dumps have been rehabilitated

Reclamation Equipment

The main technological criteria to select reclamation equipment are governed by the following main factors:

- Simplicity, high mobility, capable of doing both the job of production as well as reclamation, lower capital cost, lower maintenance and operating cost, longer life, easy to operate and maintain,

- Suitability in all operating condition.
- High degree of operational safety,
- They must be suitable to the specific mining, geological and topographical condition,
- They should be capable to meet the future demand.

Presently the reclamation works are done by the various mining equipment which are depicted as follows-

Crawler track mounted bulldozer

The dozer generally moves the spoil towards the pit in a series of cuts and the backfilling attains a final slope varying from 0° to 15°. This is very much economic for handling the spoil dump for a traveling radius of 50m to 70 m.

Conventional dragline

The dragline when is used for reclamation purpose, is stationed over the level of spoil dump created by the dozer. The boom length is generally longer varying from 50m to 94 m to operate in the maximum dumping radius to backfill the void by rehandling the spoil dump. A dozer should be employed along with the dragline so that the former pushes and accumulates the muck pile of the spoil dump within the reach of the dragline.

The muck pile will be cleaned by the dragline to fill up the void. The dozer is also utilized to doze the loose spoil which is beyond the reach of the dragline and also it does the final grading. The dragline worked over the spoil dump should have low ground bearing pressure, high mobility, high productivity and good capability to level the ground. The dragline may also be located over the high wall side of the opencast mine and a dozer or tractor is placed over the spoil. The hoist and drag rope of the dragline in this case is very long say around 200m to 300 m. The production dragline can be employed for doing the reclamation job too. However the production buckets better be replaced by a wide bottomless self digging scraper bucket to scrape efficiently heavy load during reclamation work. A pulley block is attached with the dozer or tractor through which the hoist and drag rope passes. The empty scraper bucket moves towards the dozer or tractor and when the dragline pulls the bucket towards it, the bucket dig into the loose spoil, scrape, load, transport and throw the loaded material into the void. To shift the scraping position of the bucket, the dozer or the tractor connected with the pulley block moves. After backfilling is over, leveling, grading to the final slope (1° to 15°), etc. are done by the dozer. The system is economic provided extensive spoil dump is to be re-handled. A schematic layout of land reclamation with the tower mounted dragline has been shown in the Fig 12.1.

Articulated rubber tire mounted dozer

This is one of the latest development machine for dozing, backfilling, leveling, grading etc. during development and reclamation works. The machine is very much versatile, mobile, having high speed of operation and can handle materials (here spoil dump) for a considerable traveling distance plying over the loose spoil dump with economy. But the machine is expensive. It requires high maintenance and also it cannot work on wet and sticky floor condition. The backfilling is to be done up to the ground level and sometimes dams are erected at suitable places to arrest the spoils. Above the dumped large size boulders, a filler layer of 0.5m thick with ballasts is laid to divert the rain water. Over the filler layer a layer of earth overburden (subsoil) approximates height around 1.5m and at the top of the latter top soil (around 0.5 m thick) is to be laid. Layers of soils are laid over the properly graded (grade may vary from 0° to more than 15°) backfills in chronological sequence so that the reclaimed land restore to it's original pre-mining quality condition without much of alteration. If the fragments of overburden rocks are profoundly present in the top soil, it will hinder the growth rate of the roots of the trees, adversely affect the proper distribution of moisture and ultimately it will cause detrimental affects on vegetation. Rock pickers can effectively be used to pick up the rock particles of sizes larger than 5 cm. This rock free reclaimed land will decrease the cost of tree plantation and improve the growth of vegetations. Scrapers are generally used to collect soil from the soil dump, transport it over the graded backfills and spread over them. Front end loader, tripper truck or dumper and dozer combination may also be used for reclaiming soil.

During re-grading special consideration is to be taken regarding the degree of rain fall on the reclaimed zone. If the degree of rainfall is low ,the leveled (0° grade) surface will assist rate of water absorption whereas if the degree of rainfall is heavy, the land should be graded in optimum inclination depending upon the surface topography and the climatic condition to assist good drainage of water. Both the decrease and heavily increase of water absorption prompt the degree of survival and growth rate of many verities of trees which are planted.

By:- Vikas shrimali & Vipul bansal

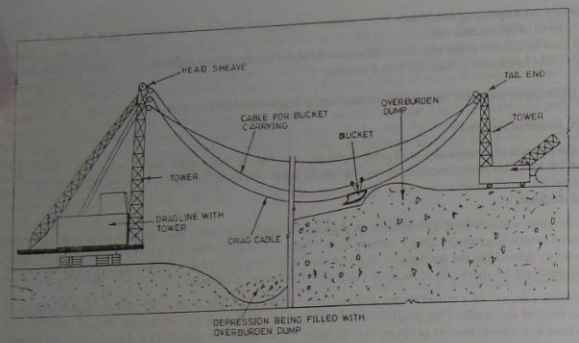


Fig. 12.1 : Land Reclamation with Tower Mounted Dragline

Grading is generally done by the bulldozers. Since the weight of the dozers are heavy due care is to be taken to prevent much of compaction of graded layers. If the soil compaction is very high , the rain water instead of absorbing internally it will escape through the outside the periphery of the reclaimed land and will cause serious problems like slope failure, erosion of high value top soil, spreading of toxic materials etc ,which will definitely cause detrimental effect of growing vegetation. Surface of the reclaimed land should be effectively sealed and stabilized. If the total excavated area cannot be backfilled, it may be converted into a beautiful lake after erecting dams in suitable places and also covering the entire exposed mineral body and the bottom of the bench with a layer of harmless cementing material. The fresh water of the lake may be used for irrigation, fishing, recreation, washing cloths, etc. If the total reclaimed land as well the high-wall slopes are nicely vegetated by the plantation of green grass, flower trees, shrubs, fruits trees, food growing plants and other species , it may be converted into a tourist spot. If any chemical changes occur, the altered land cannot be used for agricultural purpose. In that case a good soil may be borrowed and spread over it for producing crops. However, it is a very expensive process. Shopping centre, garages, community centre, school, recreation centers, etc. may also be established over the reclaimed land.

Waste Management

Waste management must be properly preplanned. When heavy amount of mineral or coal is planned to produce, heavy amount of overburden waste, washery rejects and mill tailings will also increase accordingly and they must be dumped/stacked separately from the top soil for future reclamation of excavated area. Waste dumping sites should be as near to the quarry as possible for economizing the transportation cost and as far as possible, agricultural land should be spared. Surface mine lands must be reclaimed to support pre-existing or better uses.

Wildlife

Due to opencast mining with deforestation, flora and fauna in the top soil are affected very much causing ecological imbalance. Deforestation due to mining near the forest area will cause wild animals and birds either to desert the place or die. Due to water pollution, fish may also die either in the nearby ponds or in the polluted water stream.

Wildlife agency personnel should review fish and wildlife sections to deduce the status of wildlife based habitat and vegetation structure. When potential habitats for rare or endangered species occur, more intensive monitoring program may be implemented to accommodate this need. Assemblage of relevant existing data and assessment of current permit area conditions are of great value in minimizing the long-term impact of disturbances to habitat, identifying areas of habitat for rare/endangered species and the needs of fishes and wild lives with post-mining land uses . Hence these data are to be included in the reclamation plan.

INDIAN EXAMPLE

Nigahi Project of Northern Coalfields Ltd , Chattisgarh

The old spoil dumps are round , elliptical , square or rectangular in shape . Both the old and new spoil dumps have height ranging from 10m-25m and the gradient vary between 10°-70°. Some gullies are found in them .
The depth of first coal layer varying from 30m-60m .
The overburden material consists of sandstone, boulders, pebbles, coal and other materials.
The composition of the overburden material is as follows :

Material	Percentage Composition			
	Stone	Sand	Slit	Clay
Over burden dump material	46.10	41.94	6.74	5.22
Soil				
0-15 cm	-	81.5	16.9	1.6
15-30 cm	-	80.2	18.6	1.2
30-60cm	-	75.1	23.3	1.6

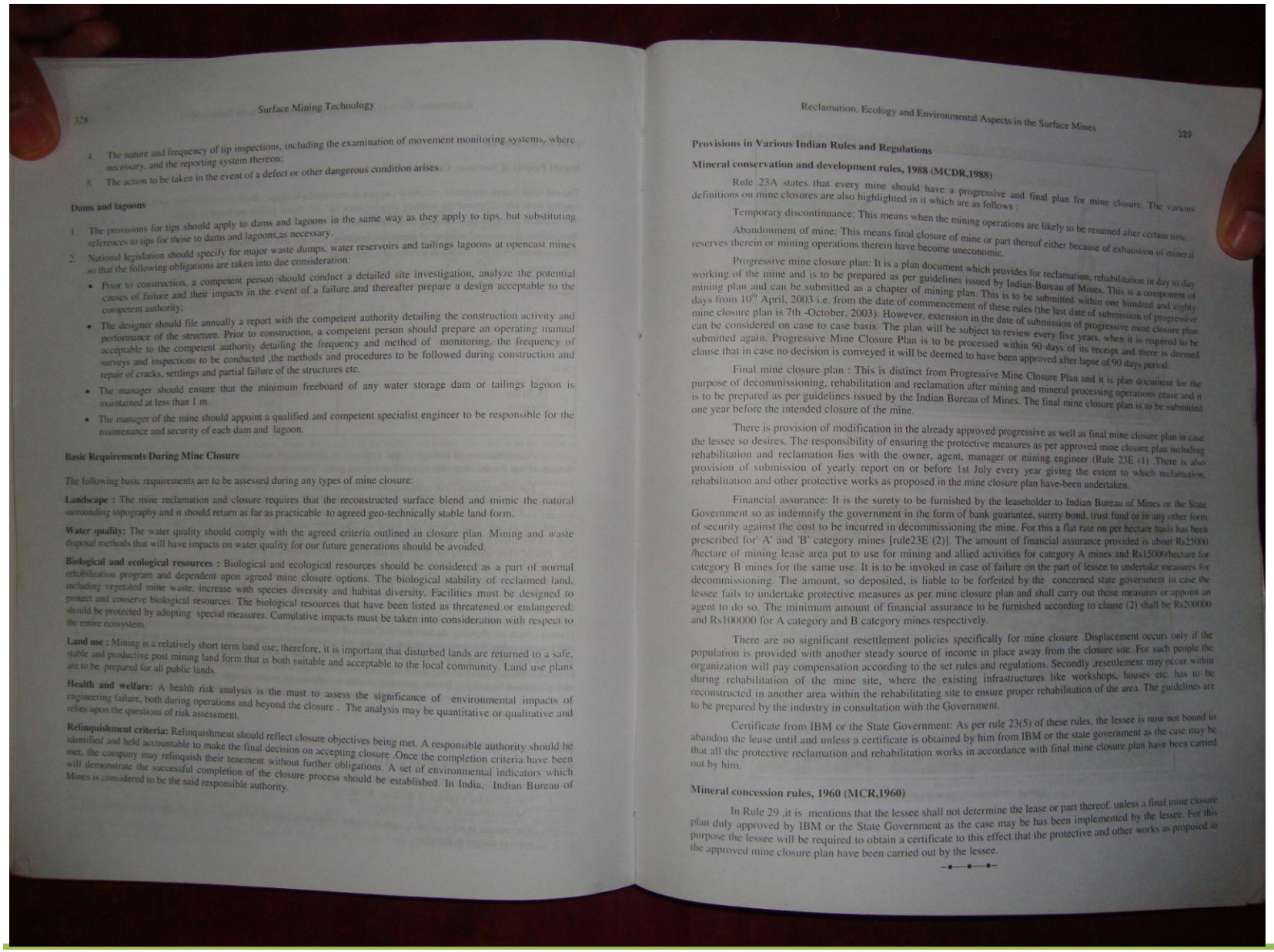
A combination of biological and engineering methods are selected for controlling of air and water erosion of mine spoil and the development of a self sustaining grass / plants through colonization of suitable species.

For afforestation , the selected overburden dumps were leveled with bulldozers . The slope of the dumps with height more than 30m-40m and steep slope was broken forming 2-3 benches of 6m-10m wide . The floor of the bench terraces and step terraces was given an inward slope so that excess rainfall would flow towards the heel .The gullies formed due to soil erosion were also filled up. Thereafter terracing was done manually along the contour of the slope selected for afforestation . Width of each terrace is 1.5m and the interval between two adjacent terraces was kept at about 2m . Pits of $45m^3$ were dug up on the top flat portion of the dump at a grid size 2 x $2m^2$. $45m^3$ pits are also dug all along the terraces at an interval of 2m. During onset of monsoon 1-1.5 years old poly-potted plants of species like Acacia auriculiformis, Acacia nilotica, Delbergia sissoo, Pongamia pinnata , Gemline arborea , Eucalyptus . Grevillea pteridifolia ,Acacia catchu , Casuarina equisetifolia , Azadarichta indica etc , were planted . Slope were also sown with the seeds of grass species like Pennisetum pedicilatum (Dinanath grass) and Stylosanthes hamata for stabilising the loose soil material and thereafter the plant species viz Gemline arborea , Eucalyptus , Grevillea pteridifolia were planted which are showing the best result. Chemical fertilizers viz. di-ammonium phosphate and urea were applied to the soil for improving the nutritional status. Regular soil working and hoeing around the plants were done for conserving the available moisture. Adequate engineering measures for soil and moisture conservation like bench terracing,construction to check dam, boulder pitching, gully plugging and erection of safety wall are taken up well before the onset of monsoon and plantation activity . Bench terracing is the mandatory to ensure proper slope. After care of plantation and introduction of fauna are being practiced for restoration of ecosystem. Main problem in opencast mining in plantation process is the dust.

ILO Recommendation

Some of the safety recommendations by ILO are as follows :
It should be the duty of the manager to prepare tipping rules which should include:

1. Provisions for a system of drainage for the tip;
2. The maintenance of drainage system.
3. The manner in which tipping operations are to be carried out in order to avoid any dangerous occurrence and prevent danger to persons;



4. The nature and frequency of tip inspections, including the examination of movement monitoring systems, where necessary, and the reporting system thereon;
5. The action to be taken in the event of a defect or other dangerous condition arises.

Dams and lagoons

1. The provisions for tips should apply to dams and lagoons in the same way as they apply to tips, but substituting references to tips for those to dams and lagoons,as necessary.
2. National legislation should specify for major waste dumps, water reservoirs and tailings lagoons at opencast mines so that the following obligations are taken into due consideration:
   - Prior to construction, a competent person should conduct a detailed site investigation, analyze the potential causes of failure and their impacts in the event of a failure and thereafter prepare a design acceptable to the competent authority;
   - The designer should file annually a report with the competent authority detailing the construction activity and performance of the structure. Prior to construction, a competent person should prepare an operating manual acceptable to the competent authority detailing the frequency and method of monitoring, the frequency of surveys and inspections to be conducted ,the methods and procedures to be followed during construction and repair of cracks, settlings and partial failure of the structures etc.
   - The manager should ensure that the minimum freeboard of any water storage dam or tailings lagoon is maintained at less than 1 m.
   - The manager of the mine should appoint a qualified and competent specialist engineer to be responsible for the maintenance and security of each dam and lagoon.

Basic Requirements During Mine Closure

The following basic requirements are to be assessed during any types of mine closure:

Landscape : The mine reclamation and closure requires that the reconstructed surface blend and mimic the natural surrounding topography and it should return as far as practicable to agreed geo-technically stable land form.

Water quality: The water quality should comply with the agreed criteria outlined in closure plan. Mining and waste disposal methods that will have impacts on water quality for our future generations should be avoided.

Biological and ecological resources : Biological and ecological resources should be considered as a part of normal rehabilitation program and dependent upon agreed mine closure options. The biological stability of reclaimed land, including vegetated mine waste, increase with species diversity and habitat diversity. Facilities must be designed to protect and conserve biological resources. The biological resources that have been listed as threatened or endangered; should be protected by adopting special measures. Cumulative impacts must be taken into consideration with respect to the entire ecosystem.

Land use : Mining is a relatively short term land use; therefore, it is important that disturbed lands are returned to a safe, stable and productive post mining land form that is both suitable and acceptable to the local community. Land use plans are to be prepared for all public lands.

Health and welfare: A health risk analysis is the must to assess the significance of environmental impacts of engineering failure, both during operations and beyond the closure . The analysis may be quantitative or qualitative and relies upon the questions of risk assessment.

Relinquishment criteria: Relinquishment should reflect closure objectives being met. A responsible authority should be identified and held accountable to make the final decision on accepting closure .Once the completion criteria have been met, the company may relinquish their tenement without further obligations. A set of environmental indicators which will demonstrate the successful completion of the closure process should be established. In India, Indian Bureau of Mines is considered to be the said responsible authority.

Provisions in Various Indian Rules and Regulations

Mineral conservation and development rules, 1988 (MCDR,1988)

Rule 23A states that every mine should have a progressive and final plan for mine closure. The various definitions on mine closures are also highlighted in it which are as follows :

Temporary discontinuance: This means when the mining operations are likely to be resumed after certain time.

Abandonment of mine: This means final closure of mine or part thereof either because of exhaustion of mineral reserves therein or mining operations therein have become uneconomic.

Progressive mine closure plan: It is a plan document which provides for reclamation, rehabilitation in day to day working of the mine and is to be prepared as per guidelines issued by Indian-Bureau of Mines. This is a component of mining plan and can be submitted as a chapter of mining plan. This is to be submitted within one hundred and eighty days from 10th April, 2003 i.e. from the date of commencement of these rules (the last date of submission of progressive mine closure plan is 7th -October, 2003). However, extension in the date of submission of progressive mine closure plan can be considered on case to case basis. The plan will be subject to review every five years, when it is required to be submitted again. Progressive Mine Closure Plan is to be processed within 90 days of its receipt and there is deemed clause that in case no decision is conveyed it will be deemed to have been approved after lapse of 90 days period.

Final mine closure plan : This is distinct from Progressive Mine Closure Plan and it is plan document for the purpose of decommissioning, rehabilitation and reclamation after mining and mineral processing operations cease and it is to be prepared as per guidelines issued by the Indian Bureau of Mines. The final mine closure plan is to be submitted one year before the intended closure of the mine.

There is provision of modification in the already approved progressive as well as final mine closure plan in case the lessee so desires. The responsibility of ensuring the protective measures as per approved mine closure plan including rehabilitation and reclamation lies with the owner, agent, manager or mining engineer (Rule 23E (1) There is also provision of submission of yearly report on or before 1st July every year giving the extent to which reclamation, rehabilitation and other protective works as proposed in the mine closure plan have-been undertaken.

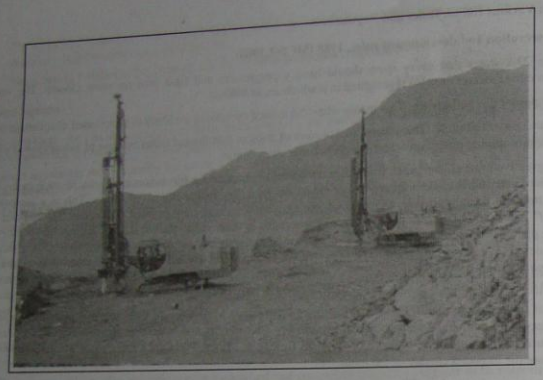
Financial assurance: It is the surety to be furnished by the leaseholder to Indian Bureau of Mines or the State Government so as indemnify the government in the form of bank guarantee, surety bond, trust fund or in any other form of security against the cost to be incurred in decommissioning the mine. For this a flat rate on per hectare basis has been prescribed for' A' and 'B' category mines [rule23E (2)]. The amount of financial assurance provided is about Rs25000 /hectare of mining lease area put to use for mining and allied activities for category A mines and Rs15000/hectare for category B mines for the same use. It is to be invoked in case of failure on the part of lessee to undertake measures for decommissioning. The amount, so deposited, is liable to be forfeited by the concerned state government in case the lessee fails to undertake protective measures as per mine closure plan and shall carry out those measures or appoint an agent to do so. The minimum amount of financial assurance to be furnished according to clause (2) shall be Rs200000 and Rs100000 for A category and B category mines respectively.

There are no significant resettlement policies specifically for mine closure .Displacement occurs only if the population is provided with another steady source of income in place away from the closure site. For such people the organization will pay compensation according to the set rules and regulations. Secondly ,resettlement may occur within during rehabilitation of the mine site, where the existing infrastructures like workshops, houses etc. has to be reconstructed in another area within the rehabilitating site to ensure proper rehabilitation of the area. The guidelines are to be prepared by the industry in consultation with the Government.

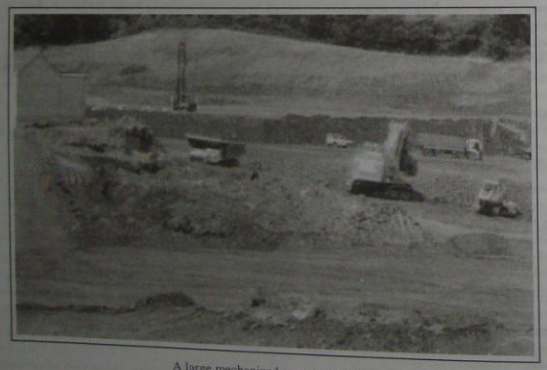
Certificate from IBM or the State Government: As per rule 23(5) of these rules, the lessee is now not bound to abandon the lease until and unless a certificate is obtained by him from IBM or the state government as the case may be that all the protective reclamation and rehabilitation works in accordance with final mine closure plan have been carried out by him.

Mineral concession rules, 1960 (MCR,1960)

In Rule 29 ,it is mentions that the lessee shall not determine the lease or part thereof. unless a final mine closure plan duly approved by IBM or the State Government as the case may be has been implemented by the lessee. For this purpose the lessee will be required to obtain a certificate to this effect that the protective and other works as proposed in the approved mine closure plan have been carried out by the lessee.



A blasthole drill under operation in a large opencast mine



A large mechanized opencast mine

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In the literary world he has already established a distinguished marks with the publication to his credit three books on his collection of classic romantic poems in Bengali named 'Jhoro Batas', 'Tumi Amari' and 'Maha Sunner Jhankar' and many more may be published in the coming future. His area of interest in the literary world is the classic romantic poetry.